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THE

METALLURGY OF GOLD.

BY

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ASSOCIATE OF THE ROYAL SCHOOL OF MINES; MEMBER OF COUNCIL OF THE INSTITUTION
OF MINING AND METALLURGY; FELLOW OF THE CHEMICAL SOCIETY;
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*BEING ONE OF A SERIES OF TREATISES ON METALLURGY,
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PREFACE TO THE FIFTH EDITION.

In the preparation of the Fifth Edition, the text has again been revised and large additions made to it, so as to keep pace with the progress of the Metallurgy of Gold, but in order to prevent the book from becoming unwieldy in size, certain portions have been omitted, either because they had become out of date, or because they were of less importance than the remainder. A new chapter on Re-Grinding has been added. The Author is indebted to the President and Council of the Institute of Civil Engineers for leave to reproduce Figs. 22 and 24, in addition to those mentioned in the text. Thanks are also due to the old Students of the Royal School of Mines and others for the information which they have sent as to work now in progress in various parts of the world.

Royal Mint, *December, 1905.*

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THE METALLURGY OF GOLD.

CHAPTER I.

THE PROPERTIES OF GOLD AND ITS ALLOYS.

PHYSICAL AND CHEMICAL PROPERTIES OF GOLD.

Introduction. — From very early times the ancients were attracted by the beautiful colour, the brilliant lustre, and the indestructibility of gold, and spared no pains in the endeavour to acquire it. In the code of Menes, who reigned in Egypt in 3600 B.C. or about 2000 years before Moses, the ratio of value between gold and silver is mentioned, one part of gold being declared equal in value to two and a half parts of silver, and it is, therefore, clear that the extraction of both metals from the deposits containing them must have been carried on before that time. It is, indeed, probable that gold was the first metal observed and collected, since it occurs in fragments of all sizes in loose sand, and the operations of collecting the larger pieces and melting them together are so simple. Among the rock carvings of Upper Egypt there are several illustrative of the art of washing auriferous sands by stirring and working them up by the hand in hollowed-out stone basins, and subsequently melting the gold in simple furnaces with the aid of mouth blow-pipes. The earliest of these carvings is supposed to date back to about 2500 B.C. However, in ancient times gold appears to have been mainly derived from India, and that country continued to supply most of the gold used in Europe until the discovery of America by Columbus.

In order to collect alluvial gold, the sands were washed down over smooth sloping rocks by means of running water, and the particles of gold, sinking to the bottom of the stream by reason of their high density, were entangled and caught in the hair of raw hides spread on the rocks. Among the hides used were sheepskins, and hence originated the form of the legend of the Golden Fleece. Stripped of its heroic dress, this legend merely describes a successful piratical expedition about 1200 B.C. to win gold, which was being laboriously obtained from streams with the help of sheepskins or goatskins by the inhabitants of what is now Armenia. Similar expeditions have not been unknown in much later times, and the method of obtaining gold by washing river sand is still practised, with improvements in matters of detail, in many parts of the world. Metallurgists are almost proverbially conservative in their methods. Hides are even now occasionally employed to catch the gold, but sheep's wool, when used, is generally in the form of blankets.

At the present day, however, when auriferous sands are washed, the aid also is invoked of what Baron Born called in 1786 the "elective affinity" of mercury for gold when mixed with impurities. The ease with which gold-amalgam can be collected, in spite of its being less dense than gold itself, is due to the fact that it is wetted by mercury.

In the history of gold, it is also of interest to the metallurgist to remember that the earliest dawn of the science of chemistry was heralded by the study of the properties of gold, and by the efforts which were made to invest other matters with these properties. From the fourth to the fifteenth century, chemistry, which was first called "chemia" (χημεια), and then "alchemy," was defined as the art of transmuting base metals into gold and silver, almost all the labours of philosophers being intended to aid directly or indirectly in solving this problem. At the end of this period, while Paracelsus was giving to chemistry a new aim—that of investigating the composition of drugs, and their effect on the human body—Agricola was reducing to order the numerous empirical facts which together made up the art of metallurgy, and although alchemy died hard, its era of usefulness may be said to have ended here. Gold has doubtless been the cause of many of the wars and marauding expeditions from which the world has suffered, but on the other hand it has been instrumental, in a far greater degree than most other commodities, in promoting the growth of civilisation, the efforts of the alchemists having laid the foundations of the science of chemistry, and those of the gold-seekers having resulted in the discovery of new countries, and in the spread of knowledge of all kinds.

Colour.—The lustre and fine colour of gold have given rise to most of the words which are used to denote it in different languages. The word "*gold*" is probably connected with the Sanscrit word "*jvalita*," which is derived from the verb "*jval*," to shine. It is the only metal which has a yellow colour when in mass and in a state of purity. Impurities greatly modify this colour, small quantities of silver lowering the tint, while copper raises it. In a finely divided state, when prepared by volatilisation or precipitation, gold assumes various colours, such as deep violet, ruby and reddish-purple, the tint varying to brownish-purple and thence to dark brown and black. This purple colour has been supposed by some experimenters (viz., Guyton de Morveau, Büchner, Desmarest, Creuzbourg and Berzelius) to be due to the formation of a coloured oxide of gold of unknown composition, but Buisson, Proust, Figuier and, more recently, Krüss have shown that no oxygen can be obtained from this coloured material, and that it probably consists of metallic gold. Similar colours are seen in purple of Cassius, and in Roberts-Austen's purple alloy of aluminium and gold, the colour in each case being probably due to a particular form of finely divided gold. "Faraday's gold," a ruby-coloured liquid prepared by the action of phosphorus and ether, or of formaldehyde on a solution of chloride of gold, is a solution of colloidal metallic gold in water.[*] Finely divided gold gives a faint blue tinge to light transmitted through the liquid in which it is suspended. Gold precipitated from its solution as bromide is in a different molecular condition from that formed from chloride, giving out 3·2 calories in passing into the latter state.[†] The surface colour of small particles of native gold is often apparently reddened by being coated with translucent films of oxides of iron. Very thin plates of gold are translucent, and appear green by transmitted light, while remaining yellow by reflected light. On heating, the green colour changes to ruby red, but is restored by the pressure of a hard substance by which the state of aggregation is again altered (Faraday). Molten gold is green, and its vapour is also probably greenish.

Malleability and Ductility.—Malleability and ductility are possessed by gold at all temperatures to a far higher degree than by any other metal. A single grain of gold can be drawn out into a wire over 500 feet long, and leaves of not more than $\frac{1}{300000}$ of an inch in thickness can be obtained by beating. Faraday has shown that the thickness of these leaves may be still further reduced by floating them in a dilute solution of potassium cyanide by which they are partly dissolved.

Hardness.—Gold is softer than silver and harder than tin. Its hardness corresponds to the number 979 in Bottone's scale, in which the diamond is 3,010.

Tenacity.—The purest gold obtainable has a tenacity of 7 tons per square inch and an elongation of 30·8 per cent., but the presence of as little as $\frac{1}{2000}$ of other elements, especially bismuth, tellurium, lead, and other metals with high atomic volumes, greatly lowers these constants, as well as the malleability and ductility of the metal, while its hardness is increased.[‡] See also the section on alloys of gold.

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[*] R. Zsigmondy, *Liebig's Annalen der Chemie*, 1898, vol. ccci., p. 29.

[†] Thomsen, *Thermochemische Untersuchungen*, vol. iii., p. 412.

[‡] Roberts-Austen, *Phil. Trans. Royal Soc.*, vol. clxxix. (1888), p. 3

Specific Gravity.—The specific gravity of gold when precipitated from solution by oxalic acid is 19·49 (G. Rose);* when cast it varies from 19·29 to 19·37, but this can be raised by compression to over 19·48.† Henry Louis has shown‡ that the specific gravity of unannealed "parted" gold (*i.e.*, the residue left after boiling silver-gold alloys in nitric acid) is 20·3, its density being lowered by the process of annealing. When precipitated by ferrous sulphate, its density may be as high as 20·72 (G. Rose).

Taking the density of pure gold at 19·3, then 1 c.c. of pure gold weighs 19·3 grammes or 0·6205 oz. troy. The weight of 1 cubic inch is 316·25 grammes or 10·168 ozs. troy. The weight of 1 cubic foot is 546·485 kilogrammes or 17569·9 ozs. troy. The volume of 1 kilogramme of gold is 51·81 c.c., or 3·162 cubic inches, and the volume of 100 ozs. troy is 161·16 c.c., or 9·835 cubic inches.

Cohesion.—On heating, gold can be welded like iron below the point of fusion, and finely divided gold agglomerates on heating without being subjected to pressure. Pressure alone is also sufficient to make gold dust cohere, while a true flow of the particles of gold can be induced in the case of the pure metal and some of its alloys.

Specific Heat.—The specific heat of gold is 0·0324 (Regnault) or 0·0316 (Violle).

Fusibility.—Gold fuses, after passing through a pasty stage, at a clear cherry-red heat, just below the fusing point of copper and much above that of silver. The metal expands considerably on fusing, and contracts again on solidifying. The freezing point was given by Heycock & Neville at 1061·7°, and more recently by Berthelot as 1064°.§

Spectrum.—In the gold spectrum Huggins saw 23 lines, the wave lengths of the most important ones being 523·1, 583·5, and 627·6 respectively.‖

Latent Heat.—The latent heat of fusion of gold is 16·3, and the normal lowering of the freezing point for 1 atom of impurity in 100 atoms of gold is 10·6°, but the presence of a few per cent. of silver causes practically no alteration in the freezing point.¶

Magnetism.—Gold is diamagnetic, its specific magnetism being 3·47 (Becquerel), if that of iron is taken as 100.

Conductivity and Expansion.—Its electrical conductivity is 76·7 and its thermal conductivity 53·2 (Wiedemann and Franz; or, according to Despretz, 103), that of silver being 100 in each case. Its coefficient of linear expansion is 0·0000144 between 0° and 100° (Fizeau).

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* *Pogg. Ann.*, vol. lxxiii. (1848), p. 1, and vol. lxxv. (1848), p. 403.

† *Eighth Report of the Royal Mint*, 1877, p. 43.

‡ *Trans. Am. Inst. of Mng. Eng.*, Chicago Meeting, 1893.

§ *Comptes Rendus*, vol. cxxvi. (1898), p. 473.

‖ For full information on the spectroscopic characteristics of gold, see Lockyer and Roberts, *Phil. Trans. Royal Soc.*, vol. clxiv. (1874), part ii., p. 495; and Frémy, *Ency. Chim.*, vol. iii. (1888), L'or, p. 40.

¶ Roberts-Austen and Rose, *Proc. Roy. Soc.*, vol. lxxi. (1902), p. 161.

Atomic Weight and Volume.—Its atomic weight, formerly believed to be about 196·2, has been more recently given by Krüss as 196·64,* by Thorpe and Laurie as 196·85,† and by Mallet as 196·79. The atomic volume of gold is 10·2.

Volatilisation of Gold.—The boiling point of pure gold has not been determined; calculated according to Wiebe's formula it would be about 2,240°, or nearly 500° above the melting point of platinum.‡ However, contrary to the belief of the older experimenters (Gasto Claveus, and others), it is sensibly volatile in air at far lower temperatures. Robert Boyle was unaware of this fact, but Homburg gilded a silver plate in 1709 by holding it over gold strongly heated in the focus of a burning mirror (*Encyclopædia Britannica*, 1778, and *Gmelin's Handbuch*), and St. Claire Deville volatilised and again condensed gold when melting it with platinum. The rapid volatilisation of gold, when heated by an ordinary blowpipe, was first proved in 1802 by Dr. Robert Hare, of Philadelphia (*Tilloch's Magazine*), a purple stain being thus produced on bone-ash in a few seconds. Lastly, a discharge of high-tension electricity from gold points causes its volatilisation, and if the discharge is sent through a fine gold wire stretched on paper, it converts it into a purple streak of finely divided condensed particles of the metal. The rapid distillation of gold caused by heating it in a current of air of considerable velocity, such as that furnished by a blowpipe, by which the liquid is thrown into waves, may be shown at any time by heating a fragment of the precious metal of the size of a pin's head on a bone-ash cupel in the oxidising flame of a good mouth blowpipe. Almost immediately after the fusion is complete, a purple stain of condensed gold begins to form on the outer margin of the cupel. The author has found that a piece of gold weighing 0·5 gramme loses half its weight in an hour, if heated on a cupel by a foot-blowpipe (the temperature attained being probably less than 1,300°), and only a few minute beads are observable, detached from the main button. Alloys of copper and gold disappear much more rapidly. No doubt most of the gold passes off as spray, but perhaps part of the loss may be due to rapid volatilisation, and could not be correctly described as mechanical loss.

The volatility of gold, both when pure and when alloyed with silver and copper, was investigated by Napier,§ who found that an alloy of 100 parts gold to 12 parts of copper, if kept for six hours at a temperature just high enough to keep it melted, lost 0·234 per cent. of its gold contents, and at the highest temperature attainable in an assay muffle, it lost 0·8 per cent. in six hours. An increase in the amount of copper present caused an increase in the loss of gold. In the simple operation of pouring about 30 lbs. of a gold-copper alloy from a graphite crucible into moulds, fumes were given off, of which the part condensed in a wet glass beaker held above the crucible contained 4·5 grains of gold. Napier also found that gold does not appear to volatilise so readily when alloyed with silver only, as when copper is also present.

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* *Berichte der deutschen chemischen Gesellschaft*, vol. xx., p. 2365.

† *Chem. Soc. Journ.*, vol. li., p. 866.

‡ L. Meyer, *Mod. Theories of Chemistry*, p. 134.

§ *Chem. Soc. Journ.*, vol. x. (1859), p. 229; and vol. xi., p. 168.

Makins found that gold volatilises sensibly along with silver and lead, when melted with these metals in a muffle in an ordinary bullion assay. The loss of gold by volatilisation on melting its copper alloy is the common experience of mints. At the Sydney mint, Leibius found that the sweepings from the top coping-stone of a chimney 70 feet high contained 1·46 per cent. of gold and 6·06 per cent. of silver. Similar results have been obtained at some other mints.

According to experiments made by the author,* the loss of gold on heating the pure metal rises with the temperature, being four times as great at 1,250° as at 1,100°, whilst it is insignificant at 1,075°, and probably *nil* at the melting point, 1,061°. The nature of the atmosphere has also an effect on the rate of volatilisation, the loss in carbon monoxide being double that in air, and six times that in coal gas. A protective layer of charcoal would, therefore, increase the loss by volatilisation. The volatilisation of gold is also increased by the presence of any metallic impurity, even by the non-volatile metals, such as platinum. Lead and platinum have a very slight effect in increasing the volatility of gold; copper and zinc a more marked effect, while 5 per cent. of antimony or mercury causes losses amounting to about 2 parts per 1,000 of gold per hour at 1,245°. The metals which have most effect in reducing the surface tension of the liquid gold appear to increase its volatility in the greatest degree.

It may be added that Hellot stated that if an alloy of 1 part of gold and 7 parts of zinc is heated in air, the whole of the gold comes off in the fumes, but recent experiments show that if mechanical loss due to the violent boiling of zinc is avoided, the amount of gold carried off by zinc fumes is insignificant.†

It has also been shown by the author‡ that tellurium does not cause volatilisation of gold at temperatures below 1,100°. Samples of an alloy of 78 per cent. of gold and 22 per cent. of tellurium were heated in a porcelain boat inclosed in a porcelain tube, through which a glass tube was passed. A current of water through the glass tube kept it cool. The alloys were heated for various lengths of time up to one hour at temperatures between 500° and 1,100°, in currents of different gases, air, carbon monoxide, hydrogen, and water gas (carbon monoxide and hydrogen in about equal volumes) being used in successive experiments.

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* *Chem. Soc. Journ.*, vol. lxiii. (1893), p. 714.

† "The Refining of Gold Bullion by Oxygen Gas." T. K. Rose. *Inst. M. and M.*, 13th April, 1905.

‡ *British Association Report*, 1897, p. 623.

In each case the whole or a part of the tellurium was sublimed and condensed on the cold tube, but the sublimates in only one case contained a trace of gold. In the other cases the whole of the gold was found still to remain in the boat. The exception was when a current of air was passed, the oxide of tellurium condensed on the cold tube in that case being found to contain 0·03 per cent. of the total gold originally present, while 99·96 per cent. of the gold was found in the boat.

A second series of experiments on a telluride ore from Western Australia, containing over 1,000 ozs. of gold per ton, gave similar results.

The losses incurred in roasting gold tellurides are probably due to fine dust being carried away mechanically, or to the absorption by the furnace bottoms of the very fusible mixtures which are formed.

Crystallisation of Gold.—Gold crystallises in the cubic system, occurring frequently in nature in the form of cubes, octahedra and rhombic dodecahedra. Cleavage is never exhibited. Single detached crystals are comparatively rare, and the crystals are usually attached end to end, forming strings, and branching, arborescent, or moss-like masses, which are composed of microscopic crystals, usually octahedra. These forms occur frequently in quartz veins, but the single crystals, which are usually of larger size—viz., from ¼ to 1½ inches in diameter — are mainly found in drift deposits. They are rarely perfect or of brilliant lustre, although such crystals were found at the Princeton Gold Mine, Mariposa County, California, but occur more frequently with rounded angles, raised edges, and cavernous faces, which are often marked with parallel striations, and possess little or no lustre (Fig. 1). The octahedra found in California are usually flattened parallel to two opposite faces, or elongated, or otherwise distorted. Still more frequently they are only partially developed, as in Figs. 2 and 3. In all these cases "the incomplete crystals have the appearance of a failure for lack of material" (W. P. Blake). Crystals of greater complexity, containing many modifying faces, occur chiefly in Siberia, Transylvania, and Brazil. The most common forms occurring naturally in

Australia are the octahedron and the rhombic dodecahedron.[*]

Liversedge found[†] that, in the majority of cases, the gold embedded in massive quartz is remarkably free from any traces of crystalline form, and the larger the fragments of gold, the less crystalline form does it present. His observations enabled him to state that all well shaped crystals of gold appear to have been formed in what are now cavities, usually left by the removal of iron pyrites, or else in very soft matrices like iron oxides, clay, calcite, and serpentine. Crystallised gold is not usually met with in the quartz of the reef itself, but in the upper portions of the ferruginous and argillaceous casing of the reef and in the detritus near its outcrop.

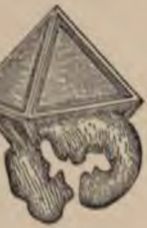

Fig. 1.

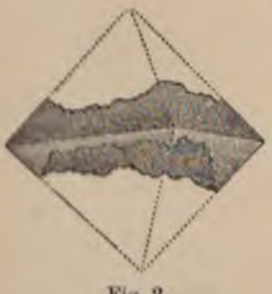

Fig. 2.

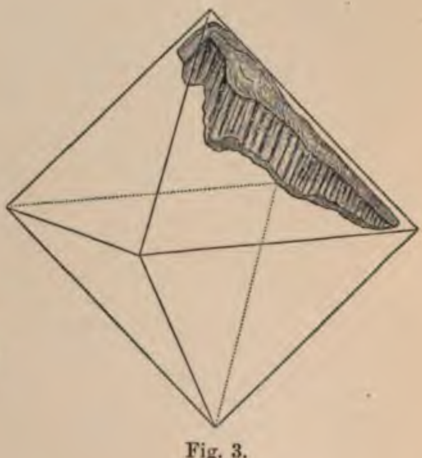

Fig. 3.

On the other hand, polished and etched sections of nuggets[‡] always show marked crystalline structure closely resembling that of a fused mass of gold. This is shown in Plate I. *A*, a section of a nugget from Coolgardie, Western Australia, which weighed 9·94 ozs., and consisted of gold 890, silver 105. Plate I. *B* shows the outside of the same nugget. There are no traces of the concentric structure which might have

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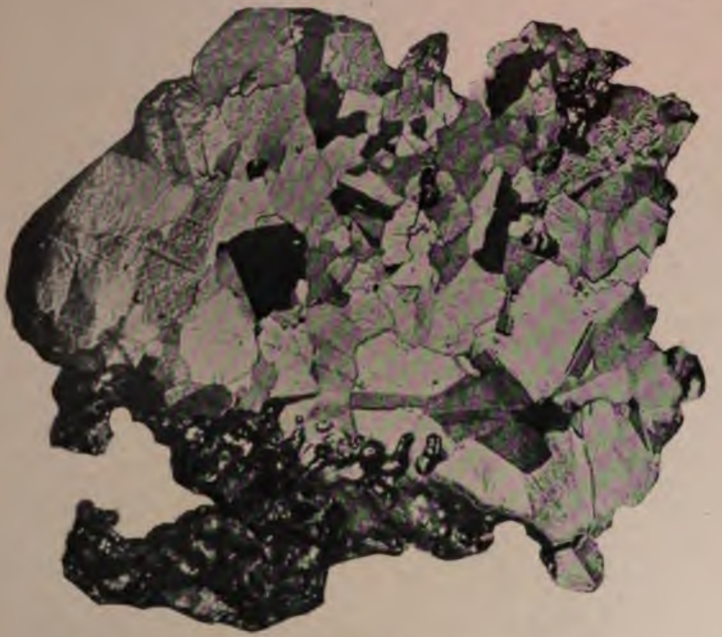
[*] For a full account of the crystalline forms of native gold, see a Paper by W. P. Blake, in *Precious Metals of the U.S.A.*, 1884, p. 573.

[†] *Journ. Roy. Soc. of New South Wales*, vol. xxvii. (1893), p. 299.

[‡] *Journ. Chem. Soc.*, vol. lxxi. (1897), p. 1125.



A. Gold Nugget, Internal Structure. × 1½ diam.

B. The same, Outside View. × 1½ diam.

been expected if the nugget had been built up of successive coatings round a nucleus.

Artificial crystals can be obtained in several ways, but with some difficulty. The slow cooling of an ingot of gold from fusion usually gives faces and sometimes angles of octahedra on the surface of the metal.* The presence of small quantities of copper reduces the size of the crystals. Feathery crystalline plates are precipitated in the electrolysis of a solution of chloride of gold and ammonium. By keeping an amalgam containing 5 per cent. of gold at a temperature of 80° for some days, and then digesting it at 30° with dilute nitric acid, bright crystals of gold can be obtained. In the Percy collection in the South Kensington Museum are some gold crystals found in the mercury troughs at the foot of the "blanket strakes," in an amalgamation mill. The troughs are placed so as to catch any stray particles of gold that may pass the blankets. As the amount of gold recovered in this way is very small, it is not worth while to clean out the troughs frequently, and in this case they had remained undisturbed for nine months, at the end of which time all amalgam was found to be crystallised. The mercury has been dissolved off by nitric acid, and the gold crystals remain. The smaller crystals are rather indefinite in shape, but amongst the larger ones (which are about half the size of a pea) are well-defined combinations of the octahedron, rhombic dodecahedron, and cube.

Solubility of Gold.—Gold is readily soluble in aqua regia, or in any other mixture producing nascent chlorine, among such mixtures being solutions of (1) nitrates, chlorides, and acid sulphates—*e.g.*, bisulphate of soda, nitrate of soda, and common salt; (2) chlorides and some sulphates—*e.g.*, ferric sulphate; (3) hydrochloric acid and nitrates, peroxides such as permanganate, or chlorates; (4) bleaching powder and acids, or salts such as bicarbonate of soda. Speaking generally, almost any chloride, bromide, or iodide will dissolve gold in presence of an oxidising agent. The action is much more rapid if heat is applied, or if the gold is alloyed with one of the base metals. The presence of silver in the gold retards the process, a scale of insoluble chloride of silver being formed over the metal, and the action may eventually be completely stopped if the percentage of silver present is large. Gold is also dissolved by liquids containing chlorine and bromine or a mixture producing bromine. The action is much slower than that of aqua regia, and subject to the same difficulties if silver is present; heat assists the dissolution. Iodine dissolves gold only if it is nascent, or if dissolved in iodides or in ether or alcohol. Metallic gold does not dissolve in strong sulphuric acid unless a little nitric acid is added when a yellow liquid is formed, which, when diluted with water, deposits the metal as a violet or brown powder. The mixture of nitric and sulphuric acids is a more rapid solvent for gold than nitric acid alone.

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* See also Chester in *Am. Journ. of Science and Arts*, vol. xvi., July, 1878, p. 29.

According to Nicklès,[*] the easily decomposable metallic perchlorides, perbromides, and periodides are capable of dissolving gold, lower chlorides, &c., of the base metals being formed, and gold chloride, &c., produced. Some of these so-called persalts are, however, often regarded merely as solutions of chlorine, bromine, or iodine in the protosalts. A boiling concentrated solution of ferric chloride readily dissolves gold. Some other haloid compounds only attack gold in the presence of ether, in which case even hydriodic acid has a slight effect. Pure hot selenic acid dissolves gold with the formation of auric selenate (Mitscherlich, also Victor Lenher). Iodic acid has also been mentioned as a solvent for gold, but its action is very slight, much less, for example, than that of concentrated hydrochloric acid under similar conditions. A mixture of iodic and sulphuric acids dissolves gold when heated to 300° (Prat, also Victor Lenher). The effect of a mixture of nitric and nitrous acids is described in Chapter xx. Alkaline sulphides attack gold slowly in the cold, and more rapidly if heated, producing sulphide of gold which is subsequently dissolved Gold is also soluble in the thiosulphates (hyposulphites) of calcium, sodium, potassium, and magnesium, in the presence of an oxidising agent.

Spring has shown[†] that gold is soluble in hydrochloric acid if heated with it to 150° in a closed tube, and is subsequently reduced by the liberated hydrogen and deposited as microscopic crystals on the side of the tube. The author has found that boiling concentrated hydrochloric acid dissolves gold and maintains it in solution. Victor Lenher showed[‡] that in presence of sulphuric acid many oxidising substances, such as telluric acid, manganese dioxide, lead dioxide, red lead, chromium trioxide, and nickelic oxide, cause gold to pass into solution. In some cases phosphoric acid may be substituted for sulphuric acid. When gold is used as an anode gold is oxidised, and if the electrolyte is strong sulphuric acid, phosphoric acid or caustic soda or potash, part of the oxide is dissolved.[§] C. Lossen pointed out in 1895 that if a solution of potassium bromide is electrolysed, the resulting alkaline solution containing hypobromite and bromate of potassium, is capable of dissolving gold. Gold is dissolved by aqueous solutions of simple cyanides in presence of an oxidising agent. Sulphocyanides, ferrocyanides, and some other double cyanides also dissolve gold, but the action is very slow, even in presence of oxidising agents, unless the solutions are heated.

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[*] *Ann. Ch. Phys.* [4], vol. x. (1867), p. 318.

[†] *Zeitschr. anorgan. Chem.*, vol. i. (1893) p. 240.

[‡] *Eng and Mng. Journ.*, June 16, 1904, p. 963, and *Journ. Amer. Chem. Soc.*, May, 1904.

[§] *Loc. cit.*, also Spiller, *Chem. News*, vol. x. (1859), p. 178.

The table on p. 12 gives the relative rate of dissolution of gold by a number of solvents. The gold used in each case consisted of a single "cornet" of "parted" gold, weighing about half a gramme, and consisting of gold 99·93 per cent., silver 0·07 per cent. Cornets offer a large surface to attack, as they consist of spongy gold, and those used did not differ in physical state. They were prepared by the method described under bullion assaying. The solutions were in excess, but no stirring or agitation was used.

Allotropic Forms of Gold.—Little is known of these. The marked influence of traces of other metals on the properties of gold has already been touched on; from this and from the variations in colour and other properties the existence of several allotropic modifications of gold might be inferred. In alloys containing appreciable quantities of other metals, evidences of allotropy are not met with so frequently. The potassium alloy, however, containing 10 per cent. of gold, on being attacked by water, leaves a black finely-divided gold powder, and there is reason to believe that this combines with water to form a hydrate.*

Wilm† states that if gold is dissolved in dilute sodium amalgam under water, the aqueous liquid becomes dark violet, and when this is acidulated with hydrochloric acid, a black precipitate of pure gold is obtained. The black gold differs from the ordinary modifications in its extreme lightness; moreover, it is soluble in alkaline solutions, and does not amalgamate with mercury or with sodium amalgam. When heated, it yields the ordinary modification as a violet red powder. This form of gold appears, from Wilm's account, to resemble the black precipitate obtained on digesting certain aluminium-gold alloys with hydrochloric acid.

ALLOYS OF GOLD.

Gold can be made to alloy with almost all other metals, but most of the bodies thus formed are of little or no practical importance. Tin, zinc, arsenic and antimony unite with gold with contraction, and form pale yellow or grey coloured, hard, brittle and easily fusible alloys, of which all, except those containing zinc, are soluble with difficulty in aqua regia. The arsenic and antimony alloys are slowly decomposed by mercury, the base metal being separated as a black powder, which consists in part of arsenide or antimonide of mercury. Lead and iron

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* Roberts-Austen, *Introd. to Study of Met.*, p. 91.

† *Zeitschr. anorgan Chem.*, vol. iv. (1893), p. 325.

SOLVENT.	Time of Action.	Temperature.	Amount of Gold dissolved per cent. per hour.
HCl and KNO_3, concentrated,	20 hours.	15° C.	4·650
HNO_3 and NaCl, ,,	,,	,,	4·311
HCl and $KClO_3$, ,,	...	,,	Completely dissolved in less than 20 hours.
HCl and $K_2Mn_2O_8$, ,,	20 hours.	,,	0·862
HCl, ,,	,,	,,	·003
,, ,,	1 hour.	70° C.	·150
HCl, 10 c.c. $K_2Mn_2O_8$, 0·2 gramme Water, 1 pint (Etard's Mixture),	20 hours. 1 hour.	15° C. 70° C.	·088 ·850
NaCl, 1·5 gramme H_2SO_4 1 c.c. $K_2Mn_2O_8$, 0·25 gramme Water, 1 pint (Black-Skeet Mixture),	20 hours. 1 hour.	15° C. 70° C.	·139 ·620
H_2SO_4, concentrated, 95 parts, HNO_3, ,, 5 ,,	20 hours. 1 hour.	15° C. 70° C.	·019 ·010
NaCl, KNO_3, and a normal sulphate, strong solution,	20 hours. 1 hour.	15° C. 70° C.	None. ,,
NaCl, KNO_3, and an acid sulphate ($KHSO_4$), strong solution,	20 hours. 1 hour.	15° C. 70° C.	,, 1·830
NaCl, and ferric sulphate, strong solution,	20 hours. 1 hour.	15° C. 70° C.	None. 0·340
HCl and H_2SO_4, concentrated,	20 hours.	15° C.	None.
Fe_2Cl_6, concentrated,	,, 1 hour.	,, 70° C.	,, 4·060
Iodine and water in closed tube,	2 hours.	50° C.	None.
Iodine dissolved in KI, strong,	20 hours.	15° C.	0·716
FeI_3 (prepared by dissolving excess of iodine in FeI_2),	,, 1 hour.	,, 70° C.	0·214 ·450
HI, pure,	20 hours. 1 hour.	15° C. 70° C.	None. ,,
,, and ether,	20 hours. 1 hour.	15° C. 70° C.	0·0007 ·460
Iodine in alcohol,	20 hours. 1 hour.	15° C. 70° C.	·001 ·120
,, in HI,	20 hours.	15° C.	4·950
Iodic acid, 5 per cent. solution,	,, 1 hour.	,, 70° C.	0·0005 ·020
Chlorine, saturated solution in water (=0·7 per cent.),	5 hours. 1¼ hours.	15° C. 60° C.	1·152 3·592
Bromine, pure,	1 hour.	15° C.	16·90
,, 5 per cent. solution in water,	,, ,,	,, 50° C.	8·30 15·71
,, 1 per cent. solution,	1¼ hours.	60° C.	5·17
,, 0·2 per cent. solution,	5 hours. ,,	15° C. 50° C.	1·162 2·020
$FeBr_3$ (prepared by dissolving iron in bromine, expelling the excess of Br by heat, dissolving in water and filtering),	20 hours. 1 hour.	15° C. 70° C.	0·003 ·070
KCy, 1 per cent. solution,	1¼ hours.	15° C.	0·15
,, 5 ,, ,,	,,	,,	·14
,, 25 ,, ,,	,,	,,	·16
,, 1 ,, ,,	,,	50° C.	·37
,, 1 ,, ,,	,,	60° C.	·46
,, 1 ,, ,,	5 hours.	100° C.	·26 Re-precipitated

alloy with gold with expansion, while in the case of copper no change of volume takes place.

Gold alloyed with a small percentage of lead is a hard, brittle, pale-yellow substance, which has a musical ring. If more than about 4 per cent. of lead is present, there is marked segregation on solidification, and this also takes place in the case of the zinc alloy and of some others.

Heycock and Neville have shown* that the freezing point of lead is lowered by the addition of gold to it in accordance with the general law. Thus, the freezing point of pure lead being 327°, an addition of 3·8 per cent. of gold reduces it to 301°, and Roberts-Austen found that the eutectic alloy of gold and lead, which contains about 13 per cent. of the former metal, melts somewhere between 190° and 198°. Similarly, by adding 6·9 per cent. of gold to thallium, the freezing point of the latter is lowered from 301° to 261°.

Several compounds of gold with other metals have been isolated, or their existence proved in various ways. Thus, for example, the compound AuSn was recognised by Matthiessen, from the curve of electric conductivity of the gold-tin series of alloys, and this substance was more recently detected by A. P. Laurie by observing the electro-motive force developed by alloys of different compositions when dipped into a solution of $SnCl_2$. Heycock and Neville succeeded in isolating the compound AuCd in 1892, after having suspected its existence for some time. In the same year, Mylius and Fromm prepared a number of gold alloys by precipitation from solution.† Thus a gold-zinc alloy, containing equal weights of the two metals, and approximating in composition to $AuZn_3$, was obtained in the form of black, spongy flocks, by adding a solution of gold sulphate to water in which a zinc plate was placed. The gold-zinc slimes obtained in the cyanide process (*q.v.*) may be compared with this. Gold-cadmium, similarly obtained, is a lead-grey crystalline precipitate, having the composition $AuCd_3$. On heating, it is converted into gold-mono-cadmium, AuCd (see above). If the gold-zinc alloy is shaken with a solution containing a cadmium salt, the gold-cadmium alloy and a zinc salt are obtained. Similarly, a copper plate, acting on solutions containing gold, yields a black, spongy compound of gold and copper; and gold-lead and gold-tin alloys in the form of black slimes are also readily prepared.

The diffusion of gold into other metals, both liquid and solid, was investigated by Roberts-Austen.‡ The rates of diffusion in liquid metals are of the same order as those of soluble salts in water but the diffusion in solids is very slow. If gold is placed at the base of a cylinder of solid lead 70 mm. high, some is found to have reached the top in thirty days, the temperature being kept at 251°. The rate of diffusion is still measurable at 100°, but is almost inappreciable at ordinary temperatures.

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* *Chem. Soc. Journ.*, vol. lxv. (1894), p. 72.

† *Chem. Soc. Journ.*, vol. lxvi., part 2 (1894), p. 236. *Nature*, vol. xliv. (1891), p. 111.

‡ *Proc. Roy. Soc.*, vol. lix., p. 281. Bakerian Lecture, *Phil. Trans.*, 1896.

The alloys which are most important to the metallurgist are those which gold forms with mercury, copper, silver, lead, and zinc.*

Gold and Aluminium.—These alloys were investigated by Roberts-Austen,† and more recently by Heycock and Neville.‡ Several true compounds of the two metals were proved to exist, the most remarkable and stable being Au_2Al, a hard white alloy with a freezing point of only 622°, and $AuAl_2$, Roberts-Austen's beautiful purple alloy which melts at a higher temperature than the fusing point of pure gold. This compound is always formed when mixtures of gold and aluminium are fused, and allowed to cool, provided that not more than about 90 per cent. of gold is present. The purple alloy is seen in patches on a white ground containing the excess of aluminium or of white compounds of gold and aluminium.

Amalgams.—A piece of gold rubbed with mercury is immediately penetrated by it and becomes exceedingly brittle. The ductility is not always restored when the mercury is removed by distillation, a crystalline structure being often induced, perhaps because part of the mercury is retained by the gold. A particle of gold "wetted" by mercury at once loses its colour. A solid amalgam is formed, but is not readily dissolved in an excess of mercury. Kasentseff has shown§ that mercury dissolves 0·11 per cent. of gold at 0°, 0·126 per cent. at 20° and 0·65 per cent. at 100°. The mercury in a mill in time becomes a saturated solution of gold, containing say 0·12 per cent. and the amalgams, which may be regarded as solutions of mercury in gold, do not mix with the mercury, but sink to the bottom if left undisturbed. By filtering, the amalgam can be removed, the liquid solution passing through the filter bag.

The composition of a saturated solution of mercury in gold is not known. It probably contains about 85 per cent. of mercury and 15 per cent. of gold. This saturated solution is solid, and is "wetted" by fresh mercury, but if more mercury is added the excess can be removed by filtering under pressure. Pasty amalgam is left in the filter bag, and becomes "drier" and less pasty the more the pressure is increased.

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* Gold is separated from its ores by lead in smelting operations, and it can be separated from molten lead by zinc or aluminium, but these processes are dealt with in other volumes of the series.

† *Proc. Roy. Soc.* vol. l. (1892), p. 367.

‡ *Phil. Trans. Roy. Soc.*, vol. cxciv. A. (1900), p. 201.

§ *Bull. Soc. Chim.* [2], vol. xxx., p. 20.

The composition of the amalgam recovered in mills in this way varies considerably. Coarse particles of gold are not saturated with mercury to their centres, although the outside layers of the particles consist of a saturated solution. The saturation of the interior, depending on the diffusion of mercury through solid amalgam, would probably not be complete for many days. The finer (*i.e.*, smaller) the particles of gold, the more nearly the interior approaches to saturation. It follows that coarse gold gives rich amalgam, and fine gold poor amalgam. The limits are pure gold on the one hand and a saturated solution of mercury in gold on the other, neither of which exist in practice. The practical limits are amalgams containing about 50 per cent. and 25 per cent. of gold respectively. These gold amalgams usually contain impurities in the shape of amalgams of silver and of base metals, as well as non-metallic substances. The actual value of squeezed amalgam consequently varies from £2 to a few shillings per ounce.

When amalgams are gradually heated, the mercury is distilled off by degrees, the action soon ceasing if the temperature is allowed to become stationary, and distillation recommencing if it is again raised. At 440° (somewhat below a red heat), an amalgam containing about three parts of gold to one of mercury is obtained, and at a bright red heat almost all the mercury is expelled, and if the heating has not been pushed too rapidly the vapours contain but little gold. The gold obstinately retains about 0·1 per cent. of mercury, which is not driven off below the melting point of gold.

Gold and Silver.—Gold and silver unite in all proportions, forming solid solutions (or isomorphous mixtures) with properties intermediate between those of the two metals. The first additions of silver do not lower the melting point of gold, and until silver forms 35 per cent. of the alloy, little change in the melting point occurs.* The first additions of gold to silver raise its melting point. The alloys are homogeneous, malleable, soft, and ductile. They are suitable for the material of trial-plates, as they are uniform in composition. The colour of gold is sensibly lowered by the addition of very small quantities of silver, and, on increasing the proportion of the latter, the colour changes by tints of a greenish-yellow (when from 20 to 40 per cent. of silver is present) to white, with a scarcely perceptible yellow tinge (when 50 per cent. of silver is present), and silver-white (when more than 60 per cent. of silver is present). The alloys containing small quantities (less than 20 per cent.) of gold are said to separate by gravity if kept for some time in a state of quiet fusion, an alloy containing one part of gold to five parts of silver ($AuAg_9$) sinking to the bottom, and slightly auriferous silver floating at the top. The silver-gold alloys most used in jewellery are *green gold* (silver 25, gold 75), *dead-leaf gold* (silver 30, gold 70), and the alloy containing 40 per cent. of silver. Triple alloys of gold, silver, and copper are employed far more frequently by English jewellers than those last mentioned; of these the alloys, consisting of 22-, 18-, 15-, 12-, and 9-carat gold respectively, can be hall marked.[*] Alloys of gold and silver were much used for coinage before the methods of parting became well known and inexpensive.

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* Roberts-Austen and Rose, *Proc. Roy. Soc.*, vol. lxxi. (1902), p. 161.

Electrum includes pale yellow alloys with from 15 to 35 per cent. of silver. It occurs native, and was much used for ornaments and coins by the Greeks and Romans, and by the nations which acquired their arts. The use of silver in the gold-copper coinage alloys was not discontinued until quite recently, all English guineas and the Australian sovereigns manufactured at Sydney up to the year 1871 containing some of it. Both nitric and sulphuric acids attack silver-gold alloys, almost completely dissolving out the silver if it is present in amounts variously stated at from 60 to 70 per cent., while, if the proportion falls below 60 per cent., some of the silver is left undissolved with the gold. Hydrochloric acid scarcely attacks these alloys, and the action of aqua regia is soon arrested if the proportion of silver is considerable. They may be dissolved by a mixture of nitric acid and a concentrated solution of common salt.

The densities of the alloys of silver and gold are as follows:—[†]

Gold.	Silver.	Specific gravity at 15°.
Per 1,000.	Per 1,000.	
1,000	0	19·26
917	83	18·08
879	121	17·54
843	157	16·96
750	250	16·03
667	333	15·07
500	500	13·60
333	667	13·00
250	750	11·78
167	833	11·28
0	1,000	10·45

Gold and Copper.—It was long believed that these metals dissolve in one another in all proportions, forming a complete series of homogeneous alloys. It has been shown, however,[‡] by an examination of the freezing point curves and the microscopic structure of the series of alloys, that they closely resemble the silver-copper series. The freezing point curves of the two series reckoned in atomic percentages are shown in Fig. 4. From this it is clear that the alloys having the lowest freezing point in each series contain 40 atoms of copper mixed with 60 atoms of gold and silver respectively. Microscopic examination proves that both these alloys are true eutectics showing the characteristic banded structure. Both are the only brittle members of their series, breaking with conchoidal fracture, and are the only homogeneous alloys of gold and copper and of silver and copper.

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[*] 22-carat gold contains $\frac{22}{24}$ by weight of fine gold, and so on. For details of these alloys see Gee's *Goldsmith's Handbook*, pp. 41-52.

[†] Hoitsema, *Zeitsch. für An. Chem.*, vol. xli. (1904), p. 66. These results agree closely with those obtained by Matthiessen.

[‡] Roberts-Austen and Rose, *Proc. Roy. Soc.*, vol. lxvii. (1900), p. 105.

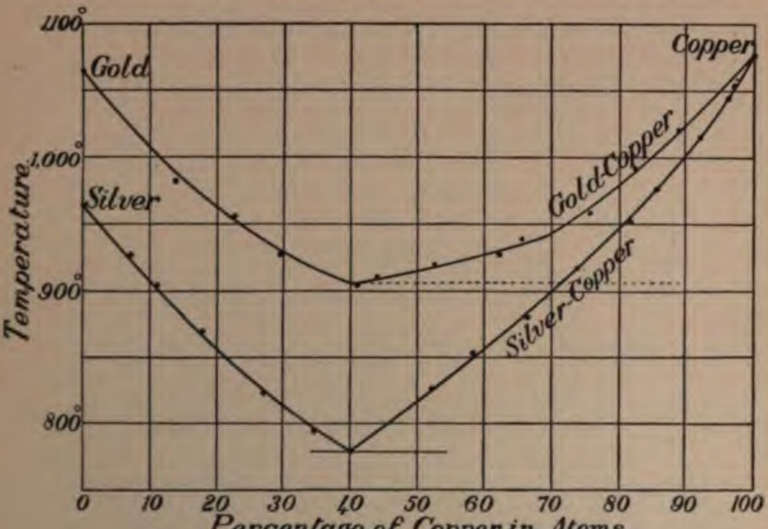

Fig. 4.

It had previously been pointed out by the author* that segregation takes place in standard gold rendered crystalline by small quantities of lead or bismuth, the presence of 0·2 per cent. of either of these metals causing the centre of a sphere 3 inches in diameter to be enriched in gold to the extent of about one part per thousand. This is due to the fact, proved by Roberts-Austen, that a eutectic alloy of gold and lead remains molten after the remainder of the mass is solidified, and is consequently driven towards the centre of the sphere, and that this fusible alloy contains much gold but very little copper. Arnold's micrographic results† tend to confirm the view which follows from this that the brittleness of crystalline gold is due to the presence of films composed of such eutectic alloys separating the crystals of gold from each other, but Osmond and Roberts-Austen* proved the non-existence of these films in quickly-cooled ingots.

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* *Chem. Soc. Journ.*, vol. lxvii. (1895), p. 552.

† *Engineering*, vol. lxi. (1896), p. 176.

Standard gold, however, is made too brittle for coinage by the presence of a far smaller amount of lead than 0·2 per cent. The addition of 0·05 per cent. of lead produces bars almost as brittle as those containing 0·2 per cent., and from researches in the Royal Mint, it would appear that 0·01 per cent. of lead or even less renders gold unfit for coinage.

The alloys of gold and copper are less malleable, harder, and more elastic than gold, and possess a reddish tint. Those with less than 12 per cent. of copper are fairly malleable; when more than this is present they are difficult to work owing to their hardness. Since no change of volume occurs when these alloys are formed, their densities may be calculated from those of gold and copper. The densities of gold-copper alloys are as follows:—†

Gold.	Copper.	Specific gravity at 15°.
Per 1,000.	Per 1,000.	
1,000	0	19·26
917	83	17·35
833	167	15·86
750	250	14·74
583	417	12·69
250	750	10·035
0	1,000	8·7

The density of standard gold is usually given as 17·48, and that of the alloy containing gold 900, copper 100 as 17·16. For the densities of other gold-copper alloys, see Rigg in the *Report of the Mint*, 1876, p. 46. Many of the alloys have been used for coinage at various times. The Greeks and Romans, after electrum had fallen into disuse, employed the purest gold they could procure, viz., that from 990 to 997 fine. Under the Roman Emperors, however, copper was intentionally added, and in the two centuries preceding the fall of Rome very base alloys were used, some containing only 2 per cent. of gold or even less.‡ In the middle ages these base alloys were discarded, and the "byzant" of Constantinople and the "florin" of Florence were both nearly pure gold, while the first gold coins struck by the nations of Western Europe were also intended to be absolutely fine. The standard 916·6 or $\frac{11}{12}$ (*i.e.*, 916·6 parts of gold in 1,000) was adopted by England in the year 1526, the standard of 994·8, which had been introduced in 1343, being finally abandoned in 1637; the 900 standard was introduced in France in 1794, and subsequently adopted in other countries. These two standards are now those most commonly used, the English standard being employed by Russia, Portugal, India, and Turkey, and the French standard by most other civilised countries; the Austrian ducat, however, has a fineness of 986 and that of Holland a fineness of 983, while the Egyptian standard is only 875. Of all these alloys the 900 and 916·6 standards are those best adapted for coinage, keeping their colour fairly well, and resisting wear better than richer alloys. The 900 alloy is harder and wears better than the 916·6 alloy, but the difference is not great, the rate of wear depending less on such small differences of composition than on the mechanical and thermal treatment of the alloys during the operation of coining.* The alloys used in coinage generally contain from two to twelve parts of silver per 1,000 in addition to the gold.

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* *Phil. Trans. Roy. Soc.*, vol. clxxxvii. (1896), A., p. 417.

† Hoitsema, *Zeit. für An. Chem.*, vol. xli. (1904), p. 65.

‡ *La Monnaie dans l'Antiquité*. Paris, 1878.

Gold-copper alloys tarnish on exposure to air owing to oxidation of the copper, and blacken on heating in air from the same cause. This oxidised coating may be removed and the colour of fine gold (not that of the original alloy) produced by plunging the metal into dilute acids or alkaline solutions, the operation being technically known as "blanching." The colour of alloys may be improved without previous oxidation by dissolving out some copper by acids, a film of pure gold being thus left on the outside which can be burnished. French jewellers use a hot solution of two parts of nitrate of potash, one part of alum, and one of common salt for this purpose.

Nitric or sulphuric acid dissolves out the copper from gold-copper alloys under conditions similar to those under which it removes silver from silver-gold alloys. If the copper falls below 6·5 per cent. the alloy is not attacked by these acids (Pearce). Aqua regia dissolves all the alloys completely.

Liquation of Gold Alloys.—The subject of liquation generally, including that observed in gold alloys, has been discussed in the volume introductory to this series.†

Liquation in the case of gold-copper alloys has been discussed above, p. 17. It has been shown by Edward Matthey ‡ that, when gold ingots containing members of the platinum group are cooled from a state of fusion, an alloy rich in the more fusible element (gold) falls out first, driving the less fusible constituent to the centre. Thus the assay of an outside cut of such an ingot gives a result too high in gold, sometimes by several per cent. It has long been known, moreover, that iridium and osmium become concentrated towards the bottom of the mass. The reason for this is that, at the temperature of fusion of gold, these refractory elements, either free or alloyed with gold, sink in the molten metal and are left in the state of small crystalline particles.

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* *Fourteenth Report of the Royal Mint*, 1884, p. 45 *et seq.*

† Roberts-Austen, *Introd. to the Study of Met.*, p. 72. The word "liquation" is used in this section as denoting the partial separation of the constituents of an alloy in such a way as to render the alloy non-uniform in composition.

‡ *Proc. Roy. Soc.*, vol. li. (1892), p. 447, and *Phil. Trans. Roy. Soc.*, vol. clxxxiii. (1892), A., pp. 629-652.

Matthey has more recently investigated* the question of segregation in connection with the alloys of gold, silver, lead and zinc produced in cyanide mills. He found that one such ingot weighing about 120 ozs. contained 662 parts per 1,000 of gold at the bottom corner, and only 439 parts at the top. In another case, when 16·4 per cent. of lead and 9·5 per cent. of zinc were present, the standard fineness of an ingot weighing 400 ozs., as shown by actually separating the whole of the precious metals, was, gold 614·0, silver 75·8, and its true value £1,028, while the value as deducted from the average of fourteen assays made on it (gold 576·0, silver 90) would have been only £965. Seven dip assays made on this ingot varied from 562 to 622 fine. Other cases of irregular distribution were even more remarkable. By experiments with synthetic alloys of gold and zinc, Matthey found that gold tends to liquate towards the centre of the mass, but only in a slight degree, the centre of a 3-inch sphere of an alloy containing gold 90, zinc 10, being only about 1 to 2½ parts per 1,000 richer than the outside. Lead acts similarly, but with greater effect, the centre being about 29 parts per 1,000 richer in gold than the outside when 30 per cent. of lead is present, but the combination of 15 per cent. of lead and 10 per cent. of zinc is still more powerful in causing segregation, the sphere being found to contain 657 parts of gold at the top, 785 in the centre, and 790 at the bottom, gravity thus playing a part. The addition of silver, however, if it amounts to not less than about two-thirds of the quantity of zinc and lead taken together, appears absolutely to prevent any liquation from taking place, an alloy containing approximately, gold 55, zinc 7, lead 18, silver 20, being practically homogeneous.

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* *Proc. Roy. Soc.*, vol. lx. (1896), p. 21.

CHAPTER II.

CHEMISTRY OF THE COMPOUNDS OF GOLD.

Compounds of Gold.—Gold is characterised chemically by an extreme indifference to the action of all bodies usually met with in nature. Its simpler compounds are formed with difficulty, and decompose readily, especially when heated. The result is that gold is found in nature chiefly in the metallic form, and the mineralogist has, therefore, few compounds to consider. Gold also forms complex compounds, especially with cyanogen and sulphur. These do not exhibit the ordinary reactions of gold, and are in particular not so readily reduced. The gold in them is not present as an elementary ion, but forms part of complex ions.

Gold forms two series of simple compounds, having the general formulæ AuR and AuR_3, while doubtful compounds corresponding to AuR_2, AuR_4, and AuR_5, have been declared to exist by Thomsen, Prat, Figuier, and others. These two series are denominated *aurous* and *auric* respectively.

Compounds of Gold with the Halogens.—Gold forms two series of compounds with the halogens, the general formulæ being AuR and AuR_3 respectively. Supposed compounds having the formula AuR_2 have been described, but are probably mixtures of the series denoted by AuR and AuR_3. All these bodies are very unstable, existing throughout only low ranges of temperature, whether in the dry state or in aqueous solution. The chlorides are the most easily formed and the least unstable, the bromides coming next. Their heats of formation are given in the subjoined table in calories :—

	Chloride.	Bromide.	Iodide.
AuR, solid, ...	+ 5·8	− 0·1	− 5·5
AuR_3, solid, . . .	+ 22·8	?	?
AuR_3, in solution, . .	+ 27·3	+ 5·1	?

CHLORIDES OF GOLD.

Gold Monochloride or Aurous Chloride, $AuCl$. — This salt is prepared by heating the trichloride to 185° in air for twelve hours. It is non-volatile and unaltered at ordinary temperatures and pressure by dry air, even when exposed to light, but begins to decompose at temperatures above 160°, and the decomposition is complete if it is heated at 175° to 180° for six days, or at 250° for one hour. Its density is 7·4. Water converts aurous chloride into a mixture of gold and gold trichloride. It is a citron-yellow amorphous powder.

Auro-Aurichloride, Au_2Cl_4.—A dark red compound having this composition is said, by Thomsen, to be obtained by heating finely-divided gold in a current of chlorine to 140° to 170°. According to Krüss and Lindet it is merely a mixture of AuCl and $AuCl_3$. It yields gold and $AuCl_3$ if brought in contact with water, ether, or alcohol.

Auric Chloride or Gold Trichloride, $AuCl_3$—*Preparation.*—Trichloride of gold can be prepared, according to Debray,[*] by heating finely-divided gold in a current of pure dry chlorine in a glass tube at a temperature of 300°. The chloride formed sublimes at this temperature and is deposited in the cooler part of the tube in fine red prisms and needles crystallising in the triclinic system. Krüss tried to repeat Debray's experiments without confirming his results.[†] He found that $AuCl_3$ was formed at 140° to 150°, and was converted to AuCl at about 180° in spite of the presence of an excess of chlorine. At 220° the AuCl was completely decomposed, leaving metallic gold, while up to this temperature a little $AuCl_3$ continued to be sublimed, but towards 300° all action ceased, and the gold remained unattacked by the chlorine at all higher temperatures.

The exact point at which gold ceases to be attacked by chlorine and the rate of volatilisation of the chloride are of great importance in connection with the loss of gold on roasting auriferous materials with salt.[‡] The matter has, therefore, been investigated by the author[§] with the following results:—Gold unites with chlorine if placed in the gas at atmospheric pressure at all temperatures up to a white heat, but the action is slight at a red heat. The absorption of chlorine by gold with the formation of chlorides at first increases in rapidity as the temperature rises, and reaches its maximum at about 225°. The fact that gold is attacked by chlorine, and that the chloride is subsequently volatilised at all temperatures between 180° and 1,100°, was proved by means of Deville's hot and cold tubes, which enable part of the sublimed chloride to be collected. The rate of volatilisation at various temperatures is as follows:—

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[*] *Comp. Rend.*, vol. xix., p. 985.

[†] *Berichte*, vol. xx., p. 212.

[‡] See Prof. Christy's experiments, Section on "Roasting," Chap. xi.

[§] *Chem. Soc. Journ.*, vol. lxvii. (1895), p. 881.

Temperature.	Percentage of Gold Volatilised in 30 minutes.	Temperature.	Percentage of Gold Volatilised in 30 minutes.
180°	0·007	580°	0·60
230°	0·35	590°	0·58
300°	2·32	805°	0·50
390°	1·82	965°	1·63
480°	0·88	1,100°	1·93

For purposes of comparison, it may be added that when gold is heated in air or coal gas, no gold is volatilised below 1,050°, and only about 0·02 per cent. in 30 minutes at 1,100°,* or about one-hundredth part of that volatilised in chlorine at the same temperature.

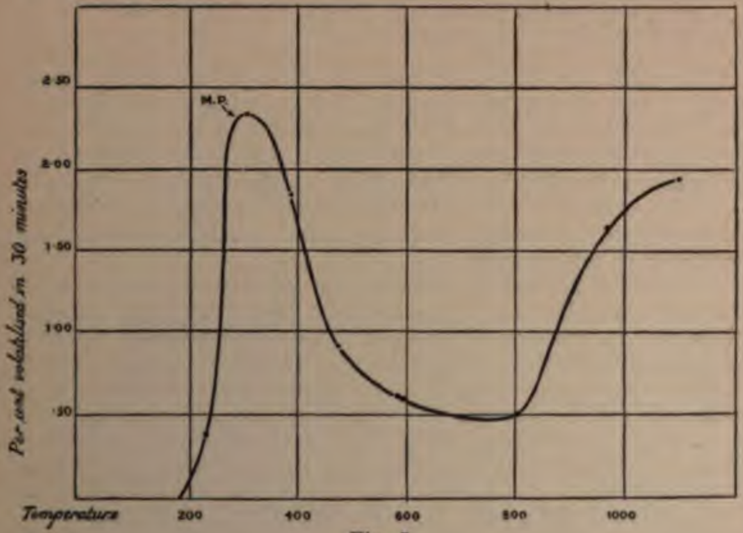

Fig. 5.

The amounts volatilised vary according to two different factors—(1) The vapour pressure of gold trichloride, $AuCl_3$, which of course increases continuously as the temperature rises; and (2) the pressure of dissociation of the trichloride, which also rises continuously with the temperature, but not at the same rate as the vapour pressure. The rise of vapour pressure tends to raise, and that of the pressure of dissociation to reduce, the amount of gold volatilised as chloride. The vapour pressure increases more rapidly than the pressure of dissociation at

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[*] *Chem. Soc. Journ.*, vol. lxiii. (1893), p. 717.

temperatures below 300°, and also above 900°, but less rapidly at intermediate temperatures. Hence the curve (Fig. 5) showing the variation of volatilisation with temperature is irregular, passing through a maximum near 300°, and a minimum at a point somewhere below the melting point of gold. The first-named change in the direction of the curve probably occurs at the melting point of the chloride—namely, 288°. The second change is perhaps caused by the change of sign of the heat of formation of the trichloride $AuCl_3$; when this becomes negative, the pressure of dissociation of the compound would decrease, in accordance with the law of van't Hoff and Le Chatelier. However this may be, it is certain that when gold is heated in chlorine at atmospheric pressure, trichloride of gold is formed and volatilised at all temperatures above 180°, up to, and probably far beyond, 1,100°.

The usual method adopted for the preparation of auric chloride is to dissolve gold in aqua regia, and then to evaporate the liquid to dryness, keeping the temperature as low as possible. A brownish-red mass is thus formed, consisting of $AuCl_3$ mixed with more or less of the protochloride and of hydrochloric acid. On taking up with water the protochloride is decomposed into gold and trichloride, but the hydrochloric acid can only be eliminated by shaking with ether, which withdraws the trichloride from its solution in water. If an attempt is made to drive off the hydrochloric acid by heat, a partial decomposition of the trichloride results.

Auric chloride exists both in the anhydrous state and in combination with two equivalents of water, $AuCl_3 . 2H_2O$, when it occurs in orange-red crystals. The anhydrous salt is of a brilliant-red colour, crystallising in needles belonging to the triclinic system, and melting at 288° under a pressure of two atmospheres of chlorine. It can be prepared by drying the hydrated salt at 150°. The anhydrous and hydrated salts are both hygroscopic, and dissolve readily in water with elevation of temperature; they are also soluble in alcohol and ether, and in some acid chlorides, such as $AsCl_3$, $SbCl_5$, $SnCl_2$, $SiCl_4$, &c.

Auric chloride is readily decomposed by heat. Lowe states* that 4 grammes of the trichloride, when heated in a porcelain basin on a boiling water bath, can be completely transformed into the monochloride, although not until after the lapse of several days. On the other hand, as has already been mentioned, Krüss states that the decomposition of auric chloride, in an atmosphere of chlorine, begins at 180°. According to the experiments of the author,† auric chloride is observed to suffer slow decomposition at as low a temperature as 165° in an atmosphere consisting of chlorine, about 1·6 per cent. being converted into monochloride in four hours at this temperature; the decomposition is about five times more rapid at 190°. The decomposition in air can be readily observed at 100°, although it does not seem to be so rapid as was indicated by Lowe. In seven days only 6·6 per cent. of the trichloride was decomposed, the initial rate of decomposition being 0·041 per cent. per hour. At 165°, however, the initial rate of decomposition appeared to be 3·2 per cent. per hour, and the conversion into monochloride was complete in four or five days at 160° and in ten hours at 190°.

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* *Dingler's polyt. Journ.*, vol. cclxxix. (1891), p. 167.

† *Chem. Soc. Journ.*, vol. lxvii. (1895), p. 902.

Solutions of gold chloride have also been shown to be decomposed by heat,* a solution of one part of the chloride in 15,000 parts of pure water yielding a precipitate of gold on heating for some hours. The solution was found to contain traces of hydrogen peroxide, the reaction being expressed by the equations—

$$AuCl_3 + 2H_2O = AuCl + 2HCl + H_2O_2.$$
$$3AuCl = AuCl_3 + 2Au.$$

A similar change, yielding hydrogen peroxide, was found by Sonstadt to take place when solutions containing 0·04 per cent. and also 0·007 per cent. of gold chloride were exposed to bright sunshine for several days.† The presence of free hydrochloric acid prevents this decomposition. Weak voltaic currents precipitate metallic gold from the solution of the trichloride upon the negative pole. The solution of trichloride of gold is also decomposed by many reducing agents, such as most organic substances, metals, and protosalts; heating the solution in every case hastens the decomposition. The reduction by organic matter is assisted by the action of light, which is especially efficacious in the presence of starchy and saccharine compounds, or of charcoal or ether. In direct sunlight the last reagent deposits a bright mirror of metallic gold, but under ordinary conditions red colloidal gold (Faraday's gold) is set free. Alkalies also quicken the action of organic matter, and it may be said that all organic compounds reduce gold chloride on boiling in the presence of potash or soda,‡ while Müller states that a mixture of glycerine and soda lye is one of the best precipitants for gold chloride, separating the metal completely in highly dilute solutions. According to Krüss, if potash and soda are quite free from organic matter, they have no action on solutions of auric chloride, whether cold or hot. If a small quantity of organic matter is present, sub-oxide of gold is precipitated; if larger quantities are present, both metallic gold and sub-oxide are precipitated in the cold, but gold alone at boiling point. Alkaline carbonates are without action on cold solutions, but if they are hot, then half the gold is precipitated as hydrate, while the other half remains in solution in the form of a double chloride of gold and the alkali. The precipitation by means of charcoal is of especial importance in view of its adoption in practice.

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* Sonstadt, *Chem. News*, vol. lxxvii. (1898), p. 74.

† *Proc. Chem. Soc.*, No. 198 (1898), p. 179.

‡ Frémy, *Ency. Chim.*, vol iii., 16e Cahier, p. 74.

It has been stated that a current of hydrogen gas will precipitate gold completely, especially on boiling the solution, but Krüss has proved that, if the hydrogen is quite pure, it has no effect either on cold or hot solutions. Sulphur, selenium, phosphorus and arsenic all precipitate gold on boiling the solution of the trichloride. Many metals reduce chloride of gold, the action being, of course, most rapid in the case of the most highly electro-positive metals, such as zinc and iron. Lead sometimes gives fine dendritic plates of gold. Sulphuretted hydrogen precipitates sulphide of gold from both neutral and acid solutions, all traces of gold being readily removed from a solution by this reagent, whilst phosphoretted, arseniuretted, and antimoniuretted hydrogen all precipitate metallic gold. The lower oxides of nitrogen, nitrous acid and many other "*ous*" acids and oxides effect the same decomposition. Sulphur dioxide is a convenient reagent, and is often used in the laboratory, being almost equally efficacious in cold and hot solutions. The reaction is

$$2AuCl_3 + 3SO_2 + 6H_2O = 2Au + 6HCl + 3H_2SO_4.$$

Various protosalts also reduce trichloride of gold. Ferrous sulphate is often used to detect the presence of gold in solution as chloride; this reagent gives the solution a pale blue colour by transmitted light, and brown by reflected light, owing to the formation of finely-divided precipitated gold. The reaction is represented by the following equation—

$$2AuCl_3 + 6FeSO_4 = 2Au + 2Fe_2(SO_4)_3 + Fe_2Cl_6.$$

To test a dilute solution for gold, a test tube filled with the liquid is held in the hand side by side with a test tube filled with distilled water and a few drops of a clear solution of ferrous sulphate are added to each. On looking down through the length of the test tubes from above, with a white surface as background, any slight changes of colour may be detected by comparison, and the liquid may also be compared with the original solution in a test tube. In this way, by a little practice, the presence of gold in the proportion of only $\frac{1}{720000}$ (1 dwt. per ton of water), or even less can be detected. The method is often used in the chlorination process, but it is better to use protochloride of tin, $SnCl_2$. This substance gives a brown precipitate of variable composition in concentrated solutions, but if mixed with the tetrachloride, $SnCl_4$, it gives a precipitate of purple of Cassius. The reaction is very sensitive, and by its means a violet coloration by transmitted light can be obtained in a solution containing 1 part of gold in 500,000 parts of water, while by special means the presence of 1 part of gold in 100,000,000 parts of water can be detected, as described below.* The liquid supposed to contain gold is raised to boiling, and poured suddenly into a large beaker containing 5 to 10 c.c. of saturated solution of stannous chloride, and the liquids agitated so as to effect complete mixture. A yellowish-white precipitate of tin hydrate forms, which settles rapidly, and can be readily separated from the bulk of the liquid by decantation. If the solution originally contained at least 1 part of gold in 5,000,000 of water (3½ grs. per ton), the precipitate is coloured purplish-red or blackish-purple, according to the nature of the solution, and the condition of the precipitant. The colour can be seen without comparing it with other precipitates. If less gold than this is present it is better to compare the precipitate with one obtained by the use of boiling distilled water, and to increase the quantity of liquid used while adhering to the same amount of stannous chloride. In this way the presence of 1 part of gold in 100,000,000 parts of water (1 grain of gold in 6 tons of water) can be detected, the amount of liquid required in this case being about 3 litres. The gold is concentrated in the precipitate in which a distinct colour is caused by less than 0·05 per cent. of the precious metal.

Chlor-auric Acid, $HAuCl_4$.—Gold trichloride in the presence of free hydrochloric acid forms this compound, which crystallises out on evaporation in vacuo in long yellow needles, having the composition $HAuCl_4 + 4H_2O$, and since gold chloride unites with many other soluble chlorides to form double chlorides, this hydrochloric acid compound is regarded as an acid. It is more stable than gold trichloride. The chlor-aurates, having the general formula $M'AuCl_4$, or $AuCl_3 . M'Cl$, are readily soluble bodies which can be crystallised, and which decompose with about the same readiness as chlor-auric acid. The sodium salt, $NaAuCl_4 . 2H_2O$, is used for "toning" in photography.

BROMIDES OF GOLD.

Gold Protobromide, $AuBr$, is a yellowish-green powder obtained by heating the tribromide to about 140°. It is insoluble in water, but is decomposed by it, metallic gold and the tribromide being formed; the change is especially rapid on boiling, and is hastened by the presence of hydrobromic acid.

Auro-auric Bromide, Au_2Br_4, is produced by the action of bromine on finely-divided gold in the cold, some tribromide being simultaneously formed. Water breaks up this bromide into $AuBr$ and $AuBr_3$, but, according to some authorities, it is only a mixture of these bodies.

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* T. K. Rose, in *Chem. News*, vol. lxvi. (1892), p. 271.

Gold Tribromide, $AuBr_3$, is produced by the action of a mixture of bromine and water on gold, particularly on the application of heat. Auric tribromide resembles the trichloride in most of its properties. It crystallises in blackish needles or scarlet plates. It is deliquescent, but is far less soluble in water than gold trichloride. Concentrated solutions which may contain about 1 per cent of the tribromide are nearly black in colour. Auric tribromide suffers decompositions similar to those noted in describing $AuCl_3$, its solutions being still less stable than those of the chloride. A solution of gold tribromide is gradually decolorised by sulphur dioxide, being completely reduced to the state of monobromide before any precipitate of metallic gold is formed. It is prepared in a pure state by heating finely-divided gold in sealed tubes with bromine and arsenic bromide, $AsBr_3$, to 126°. Gold tribromide forms intensely coloured brownish-red aqueous solutions, the presence of a mere trace of the salt in a solution being observable in this way. Double bromides exist analogous to the chlor-aurates.

The **Iodides of Gold** are of little interest to the metallurgist. They are prepared with difficulty, and decompose more readily than the chlorides and bromides, and are not likely to become of practical importance. The tri-iodide is formed if gold is acted on by a solution of iodine in potassium iodide.

CYANIDES OF GOLD.*

Cyanogen and gold unite in two proportions, forming aurous and auric cyanides, but the latter is only known with certainty in combination.

Aurous Cyanide, AuCy, is obtained by heating aurocyanide of potassium, $KAuCy_2$, with hydrochloric or nitric acid and washing with water. It is a lemon-yellow crystalline powder, insoluble in water, and unaltered by exposure to air. It is decomposed by heat, yielding metallic gold and cyanogen, and is soluble in ammonia, in yellow ammonium sulphide, in alkaline cyanides, and in hyposulphite (thiosulphate) of an alkali. It is unattacked by the mineral acids, except by aqua regia, but is decomposed when boiled with potash, metallic gold being thrown down and aurocyanide of potassium formed.

Aurocyanide of Potassium, $KAuCy_2$, is obtained by crystallisation from its solution, which is prepared by dissolving metallic gold, auric oxide or aurous cyanide in a solution of potassium cyanide. It is also formed by adding potassium cyanide to an acid solution of gold trichloride. It forms a colourless solution in water, from which it can be crystallised as colourless, transparent rhombic octahedra. Cold water dissolves 15 per cent., and boiling water twice its weight of the salt. The aqueous solution, especially if hot, gilds copper or silver without the agency of a battery. Gilding is, however, generally effected by electro-deposition, using a gold anode. Gold is also precipitated from the solution by zinc and many other metals. Precipitates are also formed on the addition of salts of zinc, copper, tin, iron, or silver, no precipitates being formed if potassium cyanide is present in excess. According to Lindbom, ferrous salts are without action on $KAuCy_2$, but oxalic acid, sulphurous acid, or mercurous chloride, Hg_2Cl_2, precipitate aurous cyanide from hot solutions.

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* See also Chapter xvi.

Aurocyanides are decomposed by mineral acids, aurous cyanide being precipitated, and hydrocyanic acid evolved. Iodine, bromine, and chlorine are dissolved by $KAuCy_2$, and the iodine compound, $KAuCy_2I_2$, can be crystallised out. The aurocyanides of sodium, ammonium, barium, calcium, zinc, cadmium, and other metals have been prepared.

Auricyanide of Potassium, $AuCy_3 . KCy$, is formed by adding potassium cyanide to a perfectly neutral solution of trichloride of gold, the precipitate first formed being redissolved. The solution is completely decolorised, and on cooling deposits colourless crystals of $AuCy_3 . KCy + 3H_2O$. These effloresce in air, giving up two molecules of water; and, on heating, the third molecule of water and some cyanogen are given off, aurocyanide of potassium being formed, and this in its turn is decomposed at a slightly higher temperature. Potassium auricyanide is somewhat soluble in cold water, and readily soluble in hot water.

OXIDES OF GOLD.

Aurous Oxide, Au_2O.—This oxide is prepared by decomposing aurous chloride, AuCl, or the corresponding bromide by potash in the cold (Berzelius) when a violet precipitate forms, which is blackish when moist, but greyish when dry. When freshly precipitated it is soluble both in alkalies and in cold water, forming an indigo blue solution, with brownish fluorescence, and on warming the solution slightly the corresponding hydrate is precipitated. It is also prepared by the action of nitrate of mercury on the trichloride, and by boiling aurate of potash with organic compounds, such as citrates or tartrates, or by boiling a solution of the trichloride with the potassium salts of these acids. When prepared according to these methods, aurous oxide always contains a certain proportion of metallic gold. The oxide may be obtained pure by reducing brom-aurate of potassium at 0° by SO_2, passing in the gas only until the solution becomes colourless, after which an excess of gas would have precipitated metallic gold. Aurous hydrate is then precipitated by potash, and, after being agglomerated by boiling, it is filtered, washed with cold water, dried, and heated to 200° to expel the water of hydration. At 250° it is resolved into gold and oxygen. Hydrochloric acid decomposes aurous oxide into metallic gold and auric salts, slowly in the cold, quickly at a boiling temperature; aqua regia dissolves the oxide, but sulphuric and nitric acids are without action on it, while weak bases at once decompose it.

An intermediate oxide, AuO, is prepared as a black powder by dissolving metallic gold in aqua regia containing an excess of hydrochloric acid, then adding an excess of carbonate of potash, and afterwards filtering and drying the precipitate. It has been little studied, but the temperature at which it decomposes has been fixed at 205° and its hydrate has been prepared.

Auric Oxide, Au_2O_3.—This, the best known oxide, is a black powder when anhydrous, and is precipitated from solutions of auric chloride in the form of a hydrate by the caustic alkalies, the carbonates of the alkalies, and hydrates of the alkaline earths or zinc. The readiest method of preparation of this compound is to add caustic potash, little by little, to a hot solution of gold chloride, until the yellow precipitate of auric hydrate, $Au(OH)_3$, first formed is dissolved to a brown liquid which contains potassium aurate, $KAuO_2$. Then a slight excess of sulphuric acid or some Glauber's salt is added, the precipitate filtered off, washed and purified from potash by being redissolved in concentrated nitric acid, and reprecipitated by dilution with water. On drying this precipitate in vacuo, the hydrate $Au_2O_3 . H_2O$, an ochreous powder, results. If it is heated to 110°, oxygen begins to be given off. At 160°, AuO remains, and on heating for some time at 250°, metallic gold remains. Trioxide of gold dissolves in concentrated sulphuric and nitric acids, from which it is partly reprecipitated on boiling or on dilution, and these solutions are supposed to contain sulphates and nitrates of gold respectively. Double nitrates of gold and the alkalies have been obtained as crystals. Hydrochloric and hydrobromic acids dissolve the trioxide forming the haloid salts, but hydroiodic acid decomposes it on boiling, giving iodine and metallic gold. Gold trioxide dissolves in boiling solutions of alkaline chlorides, giving aurates and chlor-aurates, while it also combines with metallic oxides to form aurates.

It is easily reduced by hydrogen, carbon and carbonic oxide, with the aid of very gentle heat. Boiling alcohol reduces it, yielding minute spangles of gold which were formerly used in miniature painting.

Aurates.—The aurates of potash and soda have the general formula $Au_2O_3 . R'_2O$ or $R'_2Au_2O_4$ assigned to them. They are readily soluble, crystallisable compounds, and are formed when alkalies are added in excess to solutions of gold chloride. The aurates of calcium, magnesium and zinc are insoluble in water, but soluble in hydrochloric acid.

Fulminating Gold is a compound of auric oxide with ammonia, $Au_2O_3(NH_3)_4$, which is formed by precipitating gold chloride with ammonia or its carbonate, or by the action of ammonia on gold trioxide. When prepared by the former method its composition is variable, but the fulminate is always a high explosive decomposing with violence at 145°, or on being struck, and sometimes even spontaneously. It is decomposed without explosion by sulphuretted hydrogen, and by protochloride of tin. It is a grey pulverulent powder, insoluble in water, but soluble in potassium cyanide, auricyanide of potassium being formed.

Sulphites of Gold.—Alkaline sulphites, or sulphur dioxide, which reduce gold trichloride easily, do not produce the same effect on a solution of an alkaline aurate. If sodium bisulphite is added to a boiling solution of sodium aurate ($NaAuO_2$) a yellowish precipitate is formed, soluble in excess of sodium bisulphite, and consisting of a double sulphite of gold and sodium, or *sodium aurosulphite*, having the composition $3Na_2SO_3 \cdot Au_2SO_3 + 3H_2O$. It is obtained pure by precipitating the corresponding barium salt with $BaCl_2$, and decomposing the precipitate with the minimum quantity of sodium carbonate. Double sulphites of potassium and ammonium with gold also exist. These salts are decomposed by acids, sulphite of gold being deposited, and also on boiling their aqueous solutions, but the addition of sulphuretted hydrogen or alkaline sulphides has no effect on them.

Hyposulphites of Gold.—These compounds are now called thiosulphates by chemists, but the old name is retained as being universally employed by metallurgists. They are especially interesting to the metallurgist, as on their formation depends the extraction of gold from auriferous silver ores, when these are treated by the ordinary hyposulphite, or by the Russell process. The soluble double hyposulphites of gold with the alkalies and alkaline earths have the general formula $3R''S_2O_3 \cdot Au_2S_2O_3 + 4H_2O$. The double compounds of gold with sodium, potassium, calcium, magnesium and barium are all known. The sodium salt is prepared by adding a dilute solution of gold trichloride little by little to a concentrated solution of sodium hyposulphite, when the following reaction occurs:—

$$8Na_2S_2O_3 + 2AuCl_3 = Au_2S_2O_3 \cdot 3Na_2S_2O_3 + 2Na_2S_4O_6 + 6NaCl.$$

The double hyposulphite may be separated by precipitation with strong alcohol, with which it is also washed, or it may be purified by repeated solution in water and precipitation with alcohol. Thus prepared, it consists of colourless crystalline needles, highly soluble in water, but almost insoluble in alcohol. The solution, which possesses a sweetish taste, decomposes under the influence of heat, the action being much more rapid when nitric acid is present; metallic gold and sulphate of soda are formed. Gold, however, is not reduced from its solution as double hyposulphite by either stannous chloride, ferrous sulphate or oxalic acid, although sulphuretted hydrogen and alkaline sulphides give a black precipitate of Au_2S_3. The addition of hydrochloric acid or of dilute sulphuric acid does not immediately cause an evolution of sulphur dioxide and a deposit of sulphur, as in the case of ordinary hyposulphites. Since, therefore, the double sulphite of soda and gold does not present the characteristics of either aurous salts or of hyposulphurous acid, it has been suggested that it contains a compound radical and has a composition expressed by either $Na_3S_4O_6Au$ or $Na_3S_4O_6Au + 2H_2O$. The addition of any dilute acid soon effects the decomposition of this body in solution, gold sulphide being precipitated; the reaction is accelerated by heat.

The double hyposulphites of potassium, calcium, barium and magnesium present similar characteristics. If the barium salt is treated with the amount of sulphuric acid required by theory, a solution of the acid auro-hyposulphite, $3H_2S_2O_3 . Au_2S_2O_3$, is obtained, but it cannot be crystallised. It has been supposed that the calcium salt is more easily formed than the sodium salt, and, therefore, that calcium hyposulphite is more suitable than sodium hyposulphite for use in the leaching process, whenever gold is present in perceptible quantities. According to a series of experiments conducted by Russell,* this is not the case.

Russell has demonstrated † that finely-divided gold is soluble to a limited extent (*i.e.*, 0·002 gramme in 1,000 c.c. in 48 hours), in solutions of sodium hyposulphite of all degrees of concentration. The action depends on the oxidation of the gold by the air present in the solution, the soluble double hyposulphite, $Au_2S_2O_3 . 3Na_2S_2O_3 + 4H_2O$, and caustic soda being formed.

The formation of this hyposulphite by the action of the sodium salt on gold sulphide is far more complete and rapid. In twenty-four hours at 15°, 0·066 gramme of gold, and in two hours at 65°, 0·117 gramme of gold were dissolved in dilute solutions. Since an alkaline sulphide will again precipitate Au_2S_3 from the solution, these facts seem to be at variance, and Stetefeldt suggests that the gold sulphide originally contained a small quantity of metallic gold in an excessively fine state of division, and that on heating more gold was set free. He instances Levol's statement that sulphuretted hydrogen precipitates metallic gold, and not gold sulphide, from a boiling solution of the trichloride. He believes ‡ that it is only the free gold which is attacked, and that the results obtained on attacking the sulphide were better than those in which metallic gold was used, owing to the much finer state of division in which the gold existed in the former case. Another reason for the inverse reactions may of course exist in the influence of mass, the small quantity of sodium sulphide which is formed by the reaction being insufficient to precipitate any gold, which comes down on further additions of the same reagent.

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* Stetefeldt's *Lixiviation of Silver Ores*, New York, 1888, p. 90.

† *Ibid.*, p. 19. ‡ *Ibid.*, p. 22.

SILICATES OF GOLD.

The existence of auro-silicates is now admitted without dispute, and gold has for centuries been used to impart colour to glasses, the method used being as follows:—A solution of chloride of gold is added to a mixture of sand with alkalies and alkaline earths or lead, and the whole is then fused, and colourless or yellow transparent silicates of gold thus formed. These are decomposed by being reheated gently to low redness, oxides of gold, or more probably metallic gold, being set free, and red or purple colorations thus obtained. The occurrence of silicates of gold in nature seems to be doubtful.

Experiments conducted by E. Cumenge* tend to show that the alkaline auro-silicates, obtained in the wet way, may have played an important part in the formation of auriferous quartz. The following conclusions have been established by these investigations:—

1. If an alkaline aurate, obtained by dissolving auric sesquioxide in caustic soda, is mixed with an alkaline solution of silicate of soda (soluble glass), the mixture may be concentrated by evaporation until it has attained a syrupy consistency without being decomposed. Auro-silicate of soda is, therefore, fairly stable, so long as there is an excess of alkali present.

2. The decomposition of this auro-silicate is effected by the addition of hydrochloric acid to it by which gelatinous silica is precipitated. This carries down a certain proportion of gold which gives a rose colour to the white magma.

3. This decomposition may also be completely effected by the action of an aqueous solution of carbonic acid under pressure. Thus, if the syrupy, alkaline auro-silicate is introduced into a bottle of seltzer water, which is then hermetically closed, the decomposition can be seen to be gradually going on without the semi-fluid mass being dissolved, and the latter is replaced at the end of some days by coherent silica, which, on exposure to the air, assumes a white opaline appearance tinged with rose colour.

4. When gelatinous silica, obtained by the decomposition of an alkaline auro-silicate, is heated to redness in a current of steam, it assumes either a beautiful, unalterable rose colour, or a reddish tint with visible grains of gold, according to the proportion of precious metal present, and the conditions under which the precipitation has been effected.

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* Frémy, *Ency. Chim.*, vol. iii., L'or, p. 62.

SULPHIDES OF GOLD.

These compounds are prepared as brown or black precipitates by passing sulphuretted hydrogen through a solution of gold chloride. The exact composition of the precipitate varies with the temperature and degree of concentration of the solution, and the amount of free acid present. Levol and Krüss state that Au_2S is precipitated in the cold, but that only metallic gold and free sulphur are thrown down from boiling solutions. According to others, Au_2S_3 is precipitated from cold and Au_2S from boiling solutions. It seems probable, however, that free sulphur is always formed in considerable quantities, and whether the solutions are hot or cold, dilute or concentrated, definite compounds are not precipitated, variable mixtures of the two sulphides with free sulphur and metallic gold being formed. Similar precipitates are formed by alkaline sulphides and by sulphides of most of the heavy metals. The sulphides are soluble to some extent in a saturated solution of sulphuretted hydrogen, and are easily soluble in hot solutions of alkaline sulphides or alkalies, forming double salts so that precipitation from alkaline solutions is never complete. The sulphides are readily decomposed into gold and sulphur by the action of heat, the decomposition being complete below 270°. Sulphide of gold is also dissolved at ordinary temperatures by potassium cyanide, and is slowly attacked by mercury with formation of mercury sulphide.

Purple of Cassius.—This body was discovered by Cassius of Leyden in 1683. It contains gold and oxide of tin, and is used to colour glass and glazes, various shades of violet, red and purple being thus obtainable. Several methods of preparation are used, of which the following is that employed at the factory at Sèvres*:—Half a gramme of gold is dissolved in aqua regia composed of 16·8 grammes of hydrochloric and 10·2 grammes of nitric acid, and the solution is then diluted with 14 litres of water. To this solution is added, drop by drop, a solution of a mixture of protochloride and tetrachloride of tin, prepared as follows:—3 grammes of finely-divided tin is dropped, little by little, into 18 grammes of aqua regia (constituted as above, with the addition of 5 c.c. water), the reaction is checked by cooling if it is too violent, and the solution of chloride of tin formed is allowed to cool. The precipitate of purple oxide thus obtained is finely coloured when it has been washed with boiling water. The purple precipitate obtained by Müller, by reducing chloride of gold with glucose in an alkaline solution containing tin oxide in suspension, differs from that prepared by the foregoing method in losing its colour at a red heat, while the true purple of Cassius becomes brick-red under such conditions. The true colour is seen when metallic tin acts on trichloride of gold, or when alloys of gold and tin are attacked with nitric acid.

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* Frémy's *Ency. Chim.*, vol. v., L'or, p. 63.

The composition of purple of Cassius has been the subject of much discussion. Some chemists have considered it to be a compound of aurous oxide with the oxides of tin. Debray regarded it as a lake of stannic acid coloured by finely-divided gold. Müller confirms Debray's views, showing that fine purple compounds can be made with gold and magnesia, lime, baryta, sulphate of barium, &c., the colour depending on the presence of finely-divided gold and not on the other constituent.

It was, however, shown by Schneider in 1893,* that purple of Cassius is a mixture of the hydrosols of gold and stannic acid, and fresh evidence in support of this view was given by Zsigmondy.† This investigator prepared a red solution of metallic gold in water by mixing formaldehyde rapidly with a feebly alkaline boiling solution of gold chloride. A solution of about 0·005 gramme of gold in 100 c.c. of water was thus obtained and concentrated by dialysis, until an intensely red solution containing 0·12 per cent. of colloidal gold was produced. Neutral salts, mineral acids, and alkalies precipitate the gold and an excess of alcohol changes the colour of the solution to dark violet, completely precipitating the metal, which retains the property of dissolving in water. If shaken up with mercury, these solutions are rapidly decolorised. The gold is also carried to the bottom by freshly precipitated lead sulphate and other precipitates. Gold purple of required composition and shade may be obtained by mixing solutions of colloidal gold and colloidal stannic acid and adding dilute acids or salt solutions. Precipitated purple of Cassius prepared by Zsigmondy contained after washing and ignition 40·3 per cent. of gold and 59·7 per cent. of stannic acid.

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* *Zeit. anorg. Chem.*, vol. v., p. 80.

† *Liebig's Annalen*, vol. ccci. (1898), pp .29 and 361. *Journ. Chem. Soc.*, vol. lxxiv., part 2 (1898), pp. 522 and 599.

CHAPTER III.

MODE OF OCCURRENCE AND DISTRIBUTION OF GOLD.

Dissemination of Gold.—The wide distribution of gold in minute quantities throughout the world was pointed out by W. E. Dubois, an Assayer in the United States Mint, in 1861,[*] and is further attested by a large number of specimens now in the Percy collection. These consist of small specks of gold of different sizes which have been obtained from the most varied sources. Thus, samples of Pattinson's crystallised and uncrystallised lead, pig lead from all countries, lead fume, red lead litharge, white lead, precipitated carbonate of lead and acetate of lead were all found to contain gold, which seems to be invariably present in galena. Moreover, it appears to be impossible to procure samples of copper in which gold cannot be detected, although the Lake Superior copper contains less than one part in 1,000,000; the bronze and copper coins of all nations are usually found to contain much greater quantities of gold than this. Similar evidence has been adduced which tends to show that all ores of silver, antimony and bismuth contain gold.[†]

L. Wagoner[‡] found minute quantities of gold and silver in a number of rocks taken from localities remote from veins or regions known to contain valuable minerals. Granites were found to contain an average of 0·37 part of gold per million (equivalent to over 5 grains of gold per ton), and 3·85 parts of silver per million. Sandstones contained an average of 0·03 part of gold per million, and marble 0·007 part, with larger amounts of silver. J. E. Spurr[§] drew the deduction that igneous rocks contain more gold than sedimentary rocks, and that the order of richness of the sedimentary rocks in gold and silver may, on further investigation, prove to be (1) clays and shales, (2) sandstones, (3) limestones. The gold is no doubt derived in these cases from the sea, and is concentrated where much organic matter is present. It was found many years ago at the Royal Mint that coal from the North of England contained gold.

The occurrence of gold in solution in sea water has been proved by Sonstadt,* who states that it is far less than 1 grain per ton. Liversedge subsequently showed† that the amount of gold in sea water off the coast of New South Wales is from 0·5 to 1 grain per ton, or in round numbers from 130 to 260 tons of gold per cubic mile. Dr. Don‡ could not precipitate the gold from sea water either by charcoal, by insoluble sulphides, by metals or by a current of electricity. Wagoner§ found 0·17 grain of gold per ton of water off San Francisco; and a statement has been made that about $\frac{1}{30}$ grain of gold per ton occurs in the water round the English coasts.

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[*] *Journ. Am. Phil. Soc.*, June, 1861.

[†] *Loc. cit.*, and E. A. Smith on Bismuth, &c., *Journ. Soc. Chem. Ind.*, vol. xii. (1893), p. 216.

[‡] *Trans. Am. Inst. of Mng. Eng.*, November, 1901.

[§] *Idem.*, February and May, 1902.

Gold Ores.—Gold is obtained—(1) From veins in rock formations. In this division may be included replacement-deposits, disseminations in rocks, and also, for example, the marine deposits accumulated in shallow water, such as the conglomerates of the Transvaal. (2) From placers or the alluvial deposits of ancient and modern streams. Modern beach deposits and loose sands or gravels generally, may be included in this section.

One of the most striking differences between the ores of gold and those of all other metals lies in the extremely small proportion which the desired material bears to the worthless gangue with which it is accompanied. Occasionally hand specimens in vein stuff are found containing several per cent. of the precious metal, but these are of quite exceptional occurrence and have not the slightest economic importance. The greater part of the vein gold now being produced is derived from ores containing only about one part of gold in seventy or eighty thousand, whilst, under exceptional circumstances, a yield of one part in half a million parts of gangue may give handsome profits. Placer deposits are usually much less rich than this; the average amount of gold contained in those now worked does not exceed one part in one million, and in California deposits of gravel with only one part of gold in fifteen millions have proved susceptible of successful treatment by hydraulic mining on a large scale.

(1) *Vein Gold.*—In this case the metal, whenever it is present in visible grains or masses, has sharp angular edges, although it is usually not distinctly crystalline. It frequently penetrates the rock irregularly in various directions, and is completely interwoven with, and attached to the matrix, usually quartz, so that the metal cannot be separated from the rock without crushing the latter.

The gold in lodes is sometimes in the form of crystallisations, which are, however, exceedingly rare, and crystals of gold are still probably unknown to most miners, although they occur more frequently in placer deposits. Arborescent branching and dendritic masses of crystalline gold are more common than single crystals in both quartz lodes and placer deposits. The crystalline forms met with have already been described, p. 7. In the Transylvanian lodes, gold occurs chiefly in thin sheets or plates, often as much as from half an inch to two or more inches in breadth. Such plates are rarely thicker than a visiting card, and are generally covered with crystalline lines and markings, revealing a distinct geometrical structure. Gold also occurs in wire-like forms, sometimes penetrating crystals of other minerals, such as calcite and dolomite.

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* *Chem. News*, vol. xxvi. (1872), p. 159; and *Chem. News*, March 11, 1892.

† Roy. Soc. of New South Wales, paper read on Oct. 2, 1895.

‡ *Proc. Am. Inst. Mng. Eng.*, vol. xxvii. (1897), p. 564.

§ *Loc. cit.*

It frequently happens that the gold in lodes, &c., is in a state of fine division and is not visible without magnification.

The matrix in which the gold is contained is usually quartz, but occasionally gypsum or other minerals, intersecting as veins or interlaminated with sub-crystalline, slaty, or schistose rocks, especially hydromica and chloritic slates. Gold also occurs sparingly in similar veins in granite and gneiss. It also occurs in the midst of rock formations without any obvious connection with quartz veins. Among such rocks are granite, aplite, gneiss, eurite, quartz-trachyte, syenite, andesite, basalt, diorite, gabbro, diabase, schists, porphyry and slate. These deposits seldom pay to work and in that case can hardly be called gold ores.

J. E. Spurr writes*—"Native gold has been found in both basic and acid rocks. It has also been detected in the dark ferro-magnesian silicates of rocks of all degrees of acidity. The commercially-valuable concentrations of gold are generally connected, now with basalt or gabbro, now with diorite, now with phonolite, rhyolite or granite. They occur also in many different forms, as replacement-deposits in limestone, as disseminations in igneous and sedimentary rocks, as contact-deposits near intrusive masses, and in fissure-veins."

(2) *Placer Gold and Nuggets.*—Placer gold is usually in the form of small scales, but pellets or rounded grains also occur, while large masses or nuggets are usually of a rounded mammillated form. The chief difference between the appearance of placer and vein gold lies in the fact that the former is always rounded, showing no sharp edges, even the crystals having their angles smoothed and rounded off. This has been pointed to by the advocates of the erosion theory of the origin of placer gold, as evidence in favour of their views, the roundness of the fragments being taken to prove that abrasion of the gold has been effected by attrition with water and grains of sand. The largest masses of gold yet discovered have been found in auriferous gravel. The "Blanch Barkley" nugget, found in South Aus-

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* *Trans. A.I.M.E.*, February and May, 1902, p. 21, of a paper on "Igneous Rocks as related to Occurrence of Ores."

tralia, weighed 146 pounds, and only 6 ounces of it were gangue; and one still larger, the "Welcome" nugget, from Victoria, weighed 2,195 ounces, or 183 pounds, and yielded gold to the value of £8,376 10s. 6d. In Russia a mass was found in 1842 near Miask, weighing 96 pounds troy. The largest mass from California is given in the State Mineralogist's report as weighing 2,340 ounces, or 195 pounds, but no authentic cases seem to be on record of nuggets from this State weighing more than 20 pounds.

Minerals Associated with Gold.—The minerals most common in placer deposits are platinum, iridosmine, magnetite, iron pyrites, ilmenite, zircon, garnets, rutile and barytes; wolfram, scheelite, brookite and diamonds are less common. Diamonds are associated with gold in Brazil, and also occasionally in the Urals and in the United States. In auriferous quartz lodes* the minerals most commonly associated with gold are iron and copper pyrites, zinc blende, galena, and tetradymite. Tellurides of gold are very widely distributed. Other minerals occurring with gold are tourmaline, calcite, uranium ochre, roscoelite, vanadinite, crocoite, wollastonite, gypsum, &c. The sulphides present in auriferous quartz usually contain gold; the gold in such an ore is usually in part quite free, disseminated through the quartz, in which visible grains of the metal often occur, and in part locked up in the pyrites, whence but little can in general be extracted by mercury. Dr. Don found that in many lodes in Australia, traces of gold at least were present wherever pyrites could be found and absent when no pyrites could be detected. The gold is in all probability in the metallic state in pyrites, although this is not completely established. The subject is discussed in the Section on "Gold in Pyrites," Chap. vii.

Tellurides.—Among minerals other than sulphides which contain gold, the tellurides are of most importance in metallurgy.

Calaverite is a bronze-yellow gold telluride, usually containing a little silver, occurring in Western Australia and in certain mines in California and Colorado. One analysis gave tellurium 55·5 per cent., gold 44·5 per cent., corresponding to the formula $AuTe_2$. Another analysis from Kalgoorlie showed gold 41·76 per cent., silver 0·80 per cent., and tellurium 56·65 per cent. *Krennerite* is similar in composition and appearance to calaverite but is orthorhombic. It occurs in Transylvania and Cripple Creek.† *Sylvanite*, called also *graphic tellurium*, is a telluride of gold and silver, supposed to correspond to $(AuAg)Te_2$. It sometimes contains antimony and lead in addition. It is from white to brass-yellow in colour, and the arrangement of the crystals sometimes bears a resemblance to writing characters, whence the name graphic. The following analyses agreeing in composition with sylvanite by F. C. Smith are from the Black Hills, South Dakota, and from Cripple Creek respectively*:—

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* See paper by T. A. Rickard, read before Inst. of Mining and Met., March 23, 1898.

† *Amer. Journ. of Sci.*, vol. v. (1898, iv.), p. 375.

	Au.	Ag.	Te.	Total.
I. . .	7·64	32·39	59·96	100·00
II. . .	5·61	34·23	60·16	100·00

It occurs also at Kalgoorlie, in Transylvania, in Calaveras County, California, and in Colorado.

Petzite is a telluride of silver, Ag_2Te, in which the silver is partly replaced by gold. A specimen from the Golden Rule Mine, according to Genth, contained tellurium 32·68 per cent., silver 41·86 per cent., and gold 25·60 per cent. It occurs in Transylvania, Chili, California, Colorado, and Utah.

Nagyagite or Foliated Tellurium is remarkable for being foliated like graphite, which it also resembles in its colour, a blackish lead-grey, and in having a hardness of from 1 to 1·5 only. Its density, however, is above 7. It occurs in Transylvania, and contains tellurium 32·2, lead 54·0, gold 9·0 to 13·0 per cent. sometimes with silver, copper, and sulphur in addition.

Kalgoorlite is an iron-black mineral, with sub-conchoidal fracture occurring at Kalgoorlie, Western Australia. Its analysis corresponded to the formula $HgAu_2Ag_6Te_6$.† Other gold tellurides and some native gold amalgams are occasionally met with. At Kalgoorlie, at the Cripple Creek district, Colorado, in Transylvania, in Boulder County, Colorado, and in many other localities, the value of the ore depends on the tellurides of gold contained in it.

Composition of Native Gold.—Native gold always contains silver, which occurs in varying proportions, the colour becoming paler with the increase of silver. The finest native gold yet found is that from the Pike's Peak Mine, Cripple Creek, which was 999 fine.‡ The mean fineness of Colorado gold is, however, only 800. The gold from the Mount Morgan Mine, Queensland, was formerly stated to be 997 fine, but as the gold had been extracted by chlorination, more silver may have been present originally. The finest Russian gold was that formerly obtained at Katerineburg in the Urals, and yielded gold 989·6, silver 1·6, copper 3·5, and iron 0·5 (G. Rose). Gold dust from West Africa has been found to contain 978·1 of fine gold, the remainder being silver. The gold found in the British Isles varies from 800 to 900 fine, the remainder being silver; the specimens from the district

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[*] *Trans. Am. Inst. Mng. Eng.*, vol. xxvi. (1897), p. 485.

[†] *Records Geol. Survey N.S. W.*, vol. v. (1898), p. 203.

[‡] Furman in *Colliery Manager and Metal Miner*, Oct. 1896, p. 89.

around Dolgelly are sometimes a little over 900 fine. The gold in the Andes is generally no more than 600 or 700 fine, but from the Darien Mine it is about 990.

Placer gold is usually finer than that derived from lodes, containing a smaller percentage of silver. The average composition of the placer gold formerly obtained in California was given as* gold 883·6, silver 112·4, base metals 4·0. Australian placer gold averages about 950 fine.

Gold is occasionally found alloyed with copper, and sometimes also with iron, bismuth, palladium, or rhodium. Rhodic-gold from Mexico was found to be of the specific gravity 15·5 to 16·8 and contained 34 to 43 per cent. of rhodium.

Geographical Distribution of Gold.—In the British Isles, gold is found in some of the streams of Cornwall and in lodes and river gravels near Dolgelly and in other parts of Wales, in Sutherlandshire, and near Leadhills in Scotland, and in the County of Wicklow. On the Continent of Europe, gold is most abundant in Hungary and Transylvania, where the gold occurs in quartz lodes contained in eruptive rocks of tertiary age, chiefly propylite, porphyry, diorite, and granite. The minerals occurring with the gold are galena, blende, and pyrites. In the German Empire, the gold obtained is chiefly derived from the smelting of argentiferous galena in which small quantities of the more precious metal are contained. In Italy the only important mines are those of Pestarena and Val Toppa in North Piedmont, near Monte Rosa. The pyritic ores from these mines are treated by amalgamation. Gold is also found in the sands of the Rhine, the Reuss, the Aar, and other rivers, and in small quantities in Sweden and Finland. A little gold comes from Spain (Corunna, &c.) and Servia.

The gold-bearing districts of Russia are (1) the Urals, (2) Siberia, Eastern and Western, whilst an insignificant amount is also derived from Finland and from the Caucasus. The gold was formerly derived chiefly from lodes both in the Urals and in Western Siberia, but is now almost entirely produced by the placers of Siberia.†

In India, almost all the gold now being produced is derived from the quartz lodes of the Colar goldfield, Mysore, in Southern India, in which work was begun in the year 1880. A little gold also comes from the Presidencies of Madras and Bombay. In China, Korea, and Japan considerable quantities of gold are produced; little is known of the methods used and of the amount produced in China. Some gold is obtained from the auriferous sands of Bokhara. Among other gold-producing districts of Asia and the adjoining islands may be mentioned

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* *Report of the U.S. Census*, 1880, vol. xiii., p. 352.

† See *Mineral Industry* for 1892, p. 203, and Levat, *L'or en Siberie Orientale*, Paris, 1897.

Annam and other countries in French China, the Malay Peninsula, the islands of Borneo, Celebes, Sumatra, Mindanao in the Philippines, and New Guinea.

From the United States a large percentage of the total gold production of the world is obtained. The chief producing States are California, Colorado, Dakota, Montana, Alaska, Arizona, and Utah, but smaller amounts come from many other States. The produce is now far more from lodes than from placer deposits, and in the treatment of auriferous quartz and pyritic ores almost all the known methods of treatment are applied in different localities. Canada (Yukon Territory, British Columbia, Nova Scotia and Ontario) produces large quantities of gold, and gold ores are also found in various parts of Mexico, Colombia, Bolivia, Chili, Venezuela, Brazil, Peru, and the small States of Central America. The production of several of these countries was formerly much larger than it is at the present day, the reduction being especially marked in the cases of Brazil and Venezuela.

Gold is somewhat widely distributed in Africa, the chief sources of production in former times being the placer deposits of the Gold Coast and Abyssinia. The discoveries of auriferous conglomerates in the Transvaal since 1884 have converted that region into one of the most important among gold producing countries. Rhodesia is becoming a large producer, and gold is also derived from the Gold Coast and the Ivory Coast. Egypt and Madagascar are in course of development and the Soudan is expected to yield considerable amounts in the future.

Gold is found in all the Colonies of Australia, together with Tasmania and New Zealand. In Western Australia, Kalgoorlie or East Coolgardie is by far the richest goldfield. Other important districts are Murchison, Mount Margaret, North Coolgardie, East Murchison and Coolgardie. The chief gold producing districts of Queensland are Charters Towers, Rockhamption (where the Mount Morgan mine is situated), Croydon, and Gympie. Almost all the gold is produced from the quartz mines, the placers having been practically exhausted. The chief producing districts in Victoria are Ballarat, Sandhurst (Bendigo), Beechworth, Maryborough, Castlemaine, Gippsland, and Ararat.

Origin of Gold Ores.—The origin of mineral veins, including those in which gold is contained, has long been discussed by geologists. The old theory that the quartz of veins was originally in a molten condition and was ejected from below into fissures is no longer maintained, although in 1860 H. Rosales brought forward evidence in its favour as far as the Victorian lodes are concerned. One of the theories now strongly advocated is that the materials forming the veins have been transported in aqueous solution and precipitated where they occur. In a few exceptional cases, sublimation may have played a part. The view that the solutions found their way downwards from above has been abandoned, but the *ascensional* theory and the *lateral secretion* theory both have many adherents. The last-named theory found its principal supporter during many years in Prof. F. von Sandberger, who pointed out that the gangue of many lodes varies in composition if the nature of the rocks through which they pass is changed, and claimed to have proved by analysis that the materials forming vein-stone are derived from the adjacent country rocks. He stated, moreover, that such minerals as augite, hornblende, mica, and olivine, which are essential constituents of crystalline rocks, contain small quantities of the heavy metals occurring in veins.* Although Sandberger did not try to detect gold in the silicates, this metal is not likely to be an exception. Prof. A. Stelzner objected to these conclusions, urging that small quantities of the sulphides of the heavy metals were probably mechanically mixed with the crystals of minerals which Sandberger analysed in the belief that they were pure. Stelzner advocated the retention of the ascensional theory which alone affords a satisfactory explanation of the difference in composition observable in neighbouring lodes passing through the same rocks, and apparently formed at different periods. The two theories are, however, not contradictory, and perhaps neither need be entirely rejected, the solutions being supposed to pass more or less freely in the plane of the lode after they have been impregnated.† For the origin of placer gold, see Chapter v.

In 1897, Dr. Don announced,‡ as the result of several years work in Australia and New Zealand, that the gold in those regions if present in country rock is invariably contained in pyrites. He found that the amount of pyrites and its richness in gold diminished with increase of distance from an auriferous lode, and the gold soon disappeared. In the various minerals of igneous rocks, augite, hornblende, mica, &c., no gold could be detected, although by his method the limit of accuracy was 0·15 grain per ton. He concluded, therefore, that the gold in these lodes was not derived from the country rock adjacent, but that the latter was impregnated with gold from the lode by solutions rising from a depth greater than that of any of the rocks exposed at the surface. L. Wagoner§ doubts the accuracy of these statements of Dr. Don, finding that they do not agree with his results when examining the rocks of Western America. He finds gold in rocks apparently free from sulphides.

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* *Untersuchungen über Erzgänge.* Wiesbaden, 1882 and 1885. Useful abstracts are given in Phillips' *Ore Deposits* and in Le Neve Foster's *Ore and Stone Mining.*

† See also Posepny "On the Genesis of Ore Deposits," *Trans. Am. Inst. Mng. Eng.*, 1893.

‡ "The Genesis of Certain Auriferous Lodes." *Trans. Am. Inst. Mng. Eng.*, vol. xxvii. (1897), p. 564.

§ *Trans. A.I.M.E.*, Nov., 1901.

J. E. Spurr* traces the origin of certain quartz veins direct from igneous magmas by a process of magmatic segregation, and derives an auriferous quartz vein from an igneous siliceous dyke. The magmatic concentration is succeeded by aqueous concentration acting underground by dissolution and precipitation, and on the surface chiefly by mechanical means, in both cases producing ore bodies.†

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* *Trans. A.I.M.E.*, Feb. and May, 1902.

† See also among other papers, W. H. Weed, "Ore Deposits near Igneous Contacts," *Trans. Am. Inst. Mng. Eng.*, Oct. 1902; also "Observations on Gold Deposits," by C. W. Purington, *Eng. and Mng. Journ.*, 1903, vol. l., pp. 855 and 893, where are references to other papers.

CHAPTER IV.

TREATMENT OF SHALLOW PLACER DEPOSITS.

Introduction.—The deposits grouped together under the name of "placers" comprise sands, gravels, or any loosely coherent or non-coherent alluvial beds containing gold. They have accumulated owing to the action of running water, in the beds of rivers, or on the adjoining inundation plains, or on sea beaches. They fall naturally into two groups, between which no strict line of demarcation exists. These are—

(1) Shallow or modern placers, which are usually in or near existing rivers, and have not yet been covered by other deposits. In certain arctic districts, the gravel beds have been formed by the action of glaciers.

(2) Deep level or ancient placers, which now lie buried beneath an accumulation of débris or coherent rock, the rivers by which they were formed having often been deflected into other channels by more or less extensive changes in the physical geography of the district in which they existed. Beach deposits occur in each subdivision.

In this chapter, the first of these groups will be considered.

From the earliest times to the present day, shallow placer deposits have probably yielded more gold in the aggregate than has been derived from all other sources put together. They contain metallic gold in fragments of all sizes, ranging from the finest dust to nuggets weighing thousands of ounces. Auriferous sands are found in the beds of most rivers which flow during any part of their course through a region composed of crystalline rocks. If the rivers have rocky beds, gold may be found in the crevices, caught in natural riffles, and the whole may subsequently be covered by beds of sand.

Gold also occurs in river bars and banks, in river "flats," or inundation plains, in the dry beds of streams which only flow after heavy rains ("gulch diggings"), in terrace gravels on the sides of valleys high above the present level of the water ("bench diggings"), and on the sides and tops of hills ("hill diggings"). The last two subdivisions are evidently ancient rather than modern deposits. The gravels may contain boulders of any size, up to several feet in diameter, or may shade off into fine sand, while sandy clays, especially if on the bed-rock, are frequently very rich. In the Urals, the placer deposits often consist of heavy clays, while others are formed of waterworn fragments of auriferous quartz, talcose and chloritic schists, serpentine, greenstone, &c. Gold occurs under very various conditions in these deposits. It may occur in the grass roots on inundation plains, or near the surface of the gravels in river beds, or dispersed through the whole thickness of a stratum. More commonly, however, the lowest part of the superficial beds, just above "bed-rock" (the country rock of the district), is richest. In hollows, cracks, and crevices of the bed-rock, or, if it is soft and decomposed, in the substance of the upper part of the rock itself, to the depth of 1 or 2 feet, gold occurs in the greatest quantities. In pipeclays just above bed-rock, in Victoria, it was not uncommon to find 12 ozs. of gold or more in a single tubful of "dirt," and similar rich bed-rock deposits occur in California. The depth at which bed-rock is found varies greatly; it may crop out at the surface, or it may be buried beneath hundreds of feet of gravel, and great variations occur even in a single district. In Siberia, however, the thickness of the gravel is usually less than 3 feet thick, and is rarely more than 12 feet.

Methods of obtaining Gravel from Shallow Placers.—The gravel obtained from any placer deposit is, with exceptions, ultimately all treated alike, but the mode of winning the dirt varies with the necessities of the case. On flats and bars, the surface gravel, if rich enough, is loosened with pick and shovel, and then washed. If only the part just above the bed-rock will pay for treatment, it is reached by "stripping," or, if covered by too great a thickness of barren material, shafts are sunk, and short levels run from the bottoms of them in all directions. If the nature of the ground permits of it, tunnels are run without shafts, and the rich gravel is then followed from the surface, wherever it is found. This system was much practised in the early days in California, although now seldom to be seen in operation; it was called "coyoting," from the coyote, which lives in holes in the ground. When water was encountered in the shaft, it was drawn out by a bucket, until it came in too fast, when the claim was abandoned. In California, in somewhat later times, efforts were made to reach the gold in the river beds, by deflecting the streams of water from their courses, and in other ways. These methods will be briefly described under the head "River Mining." In Siberia and in the Yukon district the gravels are perennially frozen, and are thawed with steam jets or wood fires set at the bottom of shafts and at the ends of the drifts, and are then taken out with pick and shovel.

Methods of Washing the Gravel.—*The Pan.*—When the existence of gold in the placers of California and Australia first became known, the diggers were not acquainted with any apparatus which was well adapted to extract the metal. The household pan was used everywhere to wash the gravel, and though, in its original form, it was a difficult implement to use efficiently, it has retained its place in both countries for prospecting and also for washing small quantities of rich material, however they may have been obtained. The pan is usually made of stiff sheet-iron, is flat-bottomed and circular, and at bottom 12 to 14 inches in diameter. The sides slope outwards at an angle of about 30° to the bottom, and are about 5 inches wide. A riffle is a useful addition, formed by the thickening or bulging inwards of the side, situated about half-way up the latter and running about half-way round the pan. The method of using this pan embraces several operations. First, it is filled to about two-thirds of its capacity with pay-dirt, of which it then contains from one-fifth to one-quarter of a cubic foot. It is then placed at the bottom of a water-hole or convenient stream, and the dirt is thoroughly broken up with both hands, care being taken not to leave any lumps of clay. As soon as the contents of the pan are reduced to the consistency of soft mud, the pan is grasped with both hands a little behind its greater diameter, inclined away from the operator, raised until the dirt is only just covered with water, and shaken from side to side, while a slight oscillatory circular motion is also imparted to it. The mud and fine sand are soon obtained in suspension in the water, and gradually pass over the far edge, which is lowered more and more, until little but the stones, coarse particles of sand, black sand, and gold is left. The larger stones lie on the top and are removed by hand. The final stage consists in lifting the pan with a little water in it, and by a movement of the wrist, something like that used in vanning, causing the material to be spread out by the water in a comet shape, in the angle of the pan. The "colours"—*i.e.*, yellow specks of gold—are seen at the extreme head of the comet, and also occur in the succeeding inch or two, mixed with the black sand, while the quartz-sand forms the remainder of the tail and is scraped or washed off. The gold is separated from the black sand by (*a*) amalgamation with mercury, or (*b*) drying and blowing away the black sand, a wasteful process. Liquid amalgam is readily separated from sand, and the mercury is then driven off by heat.

The Batea differs from the miner's pan in not having a flat bottom. It is of wood turned in a lathe, about 20 inches in diameter, conical, or more rarely basin shaped, and about 3 inches deep in the centre, so that the angle at the apex is about 160°. The gold collects at the lowest point and clings to the wooden surface under conditions when it would slide over iron.

The batea consequently is more rapid and effective in obtaining a "prospect" than the pan, especially when the gold is fine, but is less frequently used in the United States and Australia. The best material for the batea is mahogany cut with the direction of the grain vertical to the surface of the implement. It had its origin in South America and is especially favoured by the negro race.

Prospecting Trough.—This instrument is used in the far east, especially by the Chinese, Malays, Annamites, &c. It is made of wood, and is shaped in the form of a very flat reversed roof-top, the angle between the long sides being about 150°. In place of a circular movement of the water an alternating rocking motion is used, the water flowing up and down. The instrument is easily handled, but is very slow.

Horn Spoons cut out of black ox-horns have been used by prospectors, especially to finish the work begun by the pan. The surface holds the gold well and shows "colour" very readily.

The Cradle or *Rocker* was introduced in California soon after the first rush to the diggings took place in 1849. It consists of a rectangular wooden box, about 3 feet long and 18 inches wide, resting on two rockers (D, Fig. 6) similar to those used for infants' cradles. The shape of the walls is shown in Fig. 6, which is a section of the apparatus. The method of using it is as follows:—

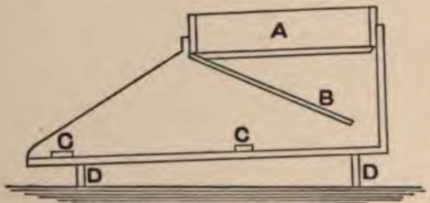

Fig. 6.
Scale 1 inch = 18 inches.

The gravel is shovelled into the riddle-box, A, the bottom of which consists of ½-inch mesh screen; the workman sits by the side of the machine and rocks it with one hand, while he pours on water by means of a dipper filled from a water-hole with the other. The dirt is disintegrated and carried through the riddle, and falls on the apron, B, which consists of blanketing, canvas or wood. Here some fine gold is caught, and the dirt then passes out from back to front over the bottom, which is slightly inclined towards the front, and the coarse gold, black sand, &c., is caught in two or three riffles, C, consisting of transverse strips of wood each of about 1 inch in height, to which mercury is sometimes added to assist in retaining the gold. The rocking motion not only assists in the disintegration of the dirt, which is effected by the water, aided by the stones, but also prevents the sand from packing behind the riffles; in the event of this happening gold would pass over the surface of the sand and be lost. Consequently the rocking should be quite continuous, since, after every pause, the sand in the riffles must be stirred up before recommencing. It is, therefore, desirable for two men to work together at the cradle, one to carry the gravel and charge it into the hopper, and to remove the large stones from the latter by hand, while the other man rocks the cradle and pours on water. It requires three or four parts of water to wash one part of gravel, and it is, therefore, better to carry the ore to water than to carry water to the ore. When a clean-up of the cradle is desirable, the riddle is removed, the apron is taken out and washed in a bucket, and the accumulations behind the riffles are scraped out and panned. Most of the fine gold in the dirt is lost by the cradle, and two men working together can only wash from 3 to 5 cubic yards per day, according to the nature of the dirt.

The Long Tom, said to have been an importation from Georgia, was first used in Nevada County, California, in the latter part of 1849.* It consists of a sluice-box or trough (A, Fig. 7) about 12 feet long, 20 inches wide at the upper end, and 30 inches at the lower end, and 9 inches deep, with an inclination of about 1 inch to the foot. The lower end of the trough is cut off at an angle of about 45° and closed by a screen of sheet-iron, B, in which a number of half-inch holes are punched, so that the fine dirt is allowed to pass through while the stones are retained. Below the screen is the upper end of the riffle-box, C, which is usually about 9 feet long, 3 feet wide, and at about the same inclination as the upper trough. It is fitted with several riffles, which are sometimes supplied with mercury. In working, a stream of water enters at the upper end of the sluice-box, into which gravel is continually shovelled, while a man breaks up the lumps with a fork, removes the large stones, and puddles the lumps of clay. Two to four men can work at one tom, and wash about five times as much in a day as can be done by one or two men with the cradle. Only the coarse gold is caught, and the machine is only suitable for washing small quantities of rich dirt where there is a plentiful supply of water. The material caught by the riffles is scraped out occasionally and panned, but the riffle-box is too short for close saving of the gold.

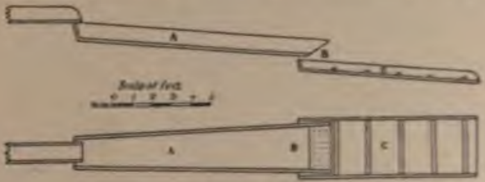

Fig. 7.

The Puddling-tub.—When water is scarce, as was the case in many places in Australia where rich gravels were found, the long-tom is inadmissible, and the puddling-tub is resorted to. This is particularly well adapted for washing clays, and is still used to disintegrate lumps of clay encountered in sluicing operations. It consists of one half of a barrel which has been sawn in two; into this the dirt is dumped and stirred up with water by means of a rake, until all the clay is held in suspension in the water, when a plug a few inches from the bottom is removed, and the slime run off. The operation is repeated until the tub is filled with gravel and sand to the level of the plug-hole and this residue is then shovelled out and washed by the pan, the cradle, or by sluicing. Large boxes were used in Australia in this way in early days, the rakes being worked by horse or steam-power; in 1860 no less than 3,858 boxes, worked by horses, were in use in Victoria alone.*

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* *Hydraulic Mining in California*, A. J. Bowie, p. 204.

The cradle, long-tom and puddling-tub are now little used in districts where work has been carried on for a long time except by the Chinese, who gain a precarious livelihood with their help by washing over the heaps of tailings accumulated from sluicing or hydraulic mining operations in Australia and California.

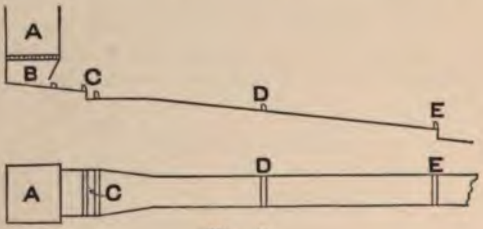

Fig. 8.

In new districts, however, for a short time after shallow deposits have been discovered, they are still extensively used. On the Klondike and at Cape Nome, for example, thousands of diggers used them, latter day improvements on the old cradle being especially numerous.

The Siberian Trough.†—In Siberia, in the Urals and in the valleys of the Obi, the Yenisei and the Lena, individual workers still exclusively use a trough, which differs from the long-tom mainly in requiring more constant attention on the part of the operator, and which resembles the old German buddle. The trough consists of a rectangular box open above and at one end.

When sandy gravels are being treated, the bottom of the disintegration box (A, Fig. 8), which is about 40 inches square, is made of a perforated screen of wood or sheet-iron, having holes of from ½ inch to 1 inch in diameter. The dirt is shovelled into this box, and, contrary to cradle-practice (see p. 48), if the gold is present in fine flakes, mercury is added here also, the amount depending on the richness of the auriferous material as determined by assay, the proportion used, however, being never more than 10 of mercury to 1 of gold. Water is directed upon the charge in the box, either by pipes from a reservoir or more often by pumping, and the fine material is carried through the screen and falls on to the table, B, while the pebbles are collected by hand and thrown away. If clay is being treated, no screen is used; the lumps are puddled in the box, and the mud carried over by an overflow of water. The table is slightly inclined, about 20 feet long, and, for the greater part of its length, is about 20 inches wide. It is furnished with five riffles, of which two (C) near the top are about 2 inches high, while the others (D, E) are of less height. The disintegration of the sand is completed on the table with the aid of a small rake continually used by the workman. When disintegration is complete, the stream of water is diminished in amount, and the workman continues to rabble the sands which have accumulated above the riffles, pushing the contents of the lower riffles up the table again, until the water runs clear, and little except pyrites is left behind the riffles where the so-called "grey concentrates" accumulate. These are either concentrated further on the same table, or removed and worked on a smaller table. In either case the stream of water is still further reduced, being graduated so as to carry away the last particles of quartz, together with all materials of moderate weight, such as garnets, rutile, tourmaline, &c., and even all the fine pyrites. If mercury has not been added previously, it is sprinkled on before this last operation, unless the gold is very coarse, when no mercury is added at any stage of the proceedings.

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* Philips' *Met. of Gold and Silver*, 1867, p. 139.

† For further particulars see the account given by Cumenge and Fuchs in Frémy's *Ency. Chim.*, vol. v., L'or, 1st Section, p. 12; and also Levat's *L'Or en Siberie Orientale*, Paris, 1897.

The "black concentrates," thus obtained, consist entirely of amalgam, magnetite, and the large grains of pyrites. The final operation, by which the amalgam is separated, is the most difficult, and requires the greatest amount of skill on the part of the operator. The material is worked on the same table with very little water, with the aid of a small rake, or more often with the hand of the workman, who kneels down by the trough for the purpose. Finally, all the pyrites having been washed away, the magnetite is removed with a magnet, and the amalgam collected. The tailings from the black concentrates are treated over again, together with the grey concentrates.

The apparatus just described treats about 500 lbs. of sand at one time, and can be worked by one man, but usually gives employment to four people (frequently three of those are women), who can treat about 5 tons of sand per day. The degree of success attained depends largely on the skill of the workman; in Siberia and Russia the art is handed down from father to son, certain families devoting their whole lives to the work during many generations. These workmen attain such a degree of dexterity in the use of the trough, that practically the whole of the valuable contents of the gravels treated are extracted by them, but the work is only suited to those who are content with small earnings.

The Sluice.—The sluice has replaced all these implements for washing the gravel from shallow placers, where water is abundant.* Sluices are constructed of "boxes," each of which resembles the upper part of the long-tom. The bottom of each box is made of rough boards, about 12 feet long, cut 4 inches wider at one end than at the other; the total width is usually from 16 to 18 inches, while the sides are 8 or 10 inches high. It has often been pointed out that sluice boxes are usually made too narrow, so that the current is too deep, and fine gold is lost, while the flow is unduly checked by the sides, thus creating undesirable eddies. The box is held together with nails, and no attempt is made to render it water-tight, as the swelling, caused by the absorption of water, and the filling up of chinks with sand and clay soon stops all leakages. The narrow end of the box fits into the wide end of that next below it, and so a sluice, made up of hundreds of boxes, can be readily put up or taken down and moved to another locality.

In all but the smallest and cheapest sluice boxes, extra strips of wood are affixed to the sides so as to protect them from the wear caused by the attrition of the stones and gravel carried through by the current. When worn thin, these strips are replaced. The bottom is similarly protected by riffle bars, whose main function is to catch the gold. Many different kinds of riffle bars are used.† The simplest are strips of cut wood or fir poles 3 inches in diameter and about 6 feet long. They are usually placed longitudinally, and are wedged in the boxes at a distance of 1 or 2 inches apart by means of transverse bars, so that two sets of riffles are placed in each box in the manner shown in Fig. 9, which represents the whole of one box and parts of two others. The depressions or riffles proper thus formed between the bars are well adapted to intercept all heavy particles that pass down the sluice, such as gold, mercury, amalgam, pyrites, &c., which gravitate to the bottom of the stream. Sometimes the riffles are placed transversely, and sometimes for a short distance in zig-zag fashion. This arrangement does not retain anything, but affords a better chance of amalgamating the gold, which, together with the mercury (in this case fed in constantly), slides down the inclined riffles from side to side of the sluice, having sunk to the bottom by virtue of its high specific gravity.

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* The large-sized sluices used in hydraulic mining are described in Chapter v.

† See *Mining Mag.*, Feb., 1905, p. 123. Article on "Washing Plants and Riffles," by C. W. Purington.

When all is ready, a stream of water is turned into the head of the sluice, where the dirt is shovelled in also. The amount of dirt shovelled in per man depends on the height of the lift and the nature of the soil, as well as on the labourer. It varies from 3 to 7 cubic yards per diem, and should average 5 or 6. The first gravel sluiced fills up most of the riffle depressions, leaving enough inequalities of surface, however, to intercept and retain the mercury, &c.

The length of the sluice varies with the consistency of the gravel, the fineness of the gold, the capital available, and the fall of the ground. It must be sufficient to complete the disintegration and then to catch the gold. The length may be adjusted by experiment; if, in the clean-up, the lowest boxes yield much amalgam, an addition to the length is necessary, while if they yield none, the sluice may be shortened by the removal of one or more boxes. Even in the latter case, however, the tailings would almost certainly contain some fine gold. The grade of the sluice is measured by inches per box, so that a grade of "12 inches" means one of 12 inches in 12 feet. The usual grade is about 6 inches per box, but it varies from 2 inches to 20 inches per box, depending—

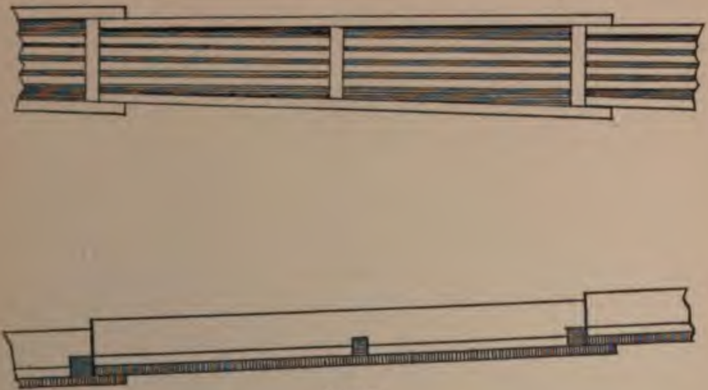

Fig. 9.

(1) *On the fall of the ground*, since the sluice cannot be raised far above it, nor sunk deep into it, owing to the increased expense thereby occasioned.

(2) *On the nature of the dirt to be washed.*—Tough, tenacious, clayey, or cemented gravels require higher grades to effect their disintegration than loose material. Instead of being disintegrated,

clay sometimes becomes aggregated into balls, which roll down the sluices, picking up particles of gold previously caught in the riffles, and these lumps of clay must be removed by hand and puddled. There must be sufficient grade to enable the water to carry away all but the largest stones, so as to avoid unnecessary hand-picking, but on the other hand, while coarse gold is readily caught, fine particles are lost if the current is too rapid.

(3) *On the quantity of water available.*—The reduction of the grade lessens the duty of the water, so that if the supply of the latter is short or costly, the grade is made as steep as possible, consistent with saving a fair proportion of the gold. A steep grade reduces the necessary length of the sluice, as disintegration takes place sooner. Since a steep grade, a rapid flow, and deep currents are best suited to effect speedy and thorough disintegration of the gravel, while a low grade, and slow and shallow currents are best adapted for saving the gold, the upper part of a sluice, for a sufficient distance to effect the complete disintegration of the gravel, is sometimes made of higher grades, or with narrower boxes than the lower part, which is occupied solely in catching the gold. When this is done additional supplies of water should be introduced at the point where the change is made, otherwise, the duty of the water being reduced, the sand packs in the angle where the grade is altered, and constant attention is required to prevent the stream from overflowing.

The requirements of disintegration and gold-saving are more often supplied by "drops," "mud-boxes," and "undercurrents." A vertical fall of the pulp constitutes a *drop*, which is arranged as follows:—The sluice terminates in a "grizzly," or inclined grating made of parallel iron bars placed longitudinally to the stream, and from 1 to 6 inches or more apart, according to the exigencies of the case. All the water and fine stuff pass through the grizzly and fall a distance of from 1 to 10 feet into a sluice below. The larger stones or boulders roll down the inclined bars, and are shot over a precipice (if possible) or on to a steep slope outside the sluice, as, unless some arrangement for removing these rocks is made, they will accumulate until they can no longer roll off the grizzly. The higher the fall, the more effectively it acts in causing disintegration. Sometimes, near the head of a sluice, the grizzly is omitted from a fall, and the boulders are retained to help in breaking up the gravel. The chief disadvantage in permitting them to remain with the rest of the gravel lies in the fact that they wear out the sluice, and that much water is required to wash them down.

A *mud-box* is merely a wide part of the sluice, 2½ or 3 feet wide, and 12 or more feet long, at which a man is stationed to break up and puddle the lumps. A better, but more expensive, appliance is a trommel or a pan with a mechanical stirrer.*

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* *Mining Mag.*, Jan., 1905, p. 16. Article by C. W. Purington.

The *undercurrent* is often used in conjunction with a drop. A grizzly with bars placed close together allows most of the water and fine material to pass through, while the coarse stuff is carried over and falls into the main sluice below. The fine material is carried off by a short sluice placed at right angles to the general direction of the main sluice, and is discharged into the upper end of a large broad box from three to ten times as wide as the sluice, and with its long diameter parallel to the main sluice. A number of check-boards help to distribute the stream evenly over the whole width of the box. This box, to which the name undercurrent is often given, although it properly belongs to the whole arrangement, is usually of higher grade than the sluice, having a fall of 10 or 15 inches per box, so that a broad shallow stream flows over its surface. It is plentifully supplied with riffles and mercury, and is intended to catch fine gold and amalgam. The tailings from the undercurrent are discharged into the main sluice below the drop.

Both grizzlies and undercurrents are used more frequently in hydraulic mining than in shallow placer sluicing, in which the large stones are usually removed by a man with a blunt pronged fork, who also either breaks up the lumps of clay or removes them and puddles them in tubs.

The Use of Mercury in Sluicing.—Mercury is added at the head of the sluice after washing has been in progress for a sufficiently long time for all leakages to have been stopped, and for the lowest depressions to have been filled in with sand. The mercury is sometimes sprinkled into the sluice, and sometimes poured into the riffles. The amount added varies with the richness of the dirt and the magnitude of the operations, enough being added to dissolve the amalgam formed. It is carried down the sluice and lodges in the riffles, the greater part being retained in the first few boxes. Fresh supplies are introduced every few hours at the head of the sluice, and sometimes at various points lower down the sluice also; in particular, mercury is added to the undercurrents, as it is especially valuable in catching the finer particles of gold which would otherwise be lost, whilst coarse gold can in great part be saved without mercury. Sometimes the latter is forced into the substance of the wooden riffles by driving an iron gas pipe into the wood, and filling it up with mercury, which is forced by the pressure of the column through the pores of the wood. The amalgam then forms on the surface and in the interstices of the blocks, and in cleaning-up this is scraped off. A better plan is to use amalgamated copper plates, which is now often done. These resemble the plates used in stamp batteries, described in Chapter vi.

The Clean-up.—The length of each "run," at the end of which the boxes are cleaned-up, varies, according to the richness of the gravel, from a day to a whole season, and is usually a week.

The upper part of the sluice, which retains most of the gold, is usually cleaned-up more frequently than the remainder. A clean-up is begun by discontinuing the supply of gravel, and letting the water continue to flow until it passes through the sluice quite clear. The first six or eight sets of riffle bars are then taken up, and the sand, mercury, and amalgam washed down, all the latter being caught by the first riffle left in. It is scooped out thence by a wooden ladle or iron spoon into a bucket, and the rich sand is collected and panned. The next few riffle bars are now taken up, and so on, or alternatively the work may be begun on several sections at the same time. Lastly, the whole sluice is carefully searched over, and particles of amalgam or mercury picked out of every crevice where they have lodged by spoons, penknives, &c.

The amalgam thus collected is stirred with fresh quicksilver in "amalgam kettles" or buckets, and the black sand and other foreign matter skimmed off. It is then strained through chamois leather or drilling, liquid mercury passing through and pasty amalgam being retained by the skin. The amalgam is well squeezed and then retorted. The retorts used in well-conducted enterprises are similar to those in use in stamp mills, described in Chapter vii. Amalgam obtained as the result of operations on a small scale, however, is often merely heated on a shovel over an ordinary fire, the mercury being driven off and lost.

Tail Race.—The tailings from sluicing operations on low ground which has not much fall are removed through a covered-in wooden sluice, or, better still, through a large iron pipe. The work proceeds in the up-stream direction, and the worthless material stripped from above the pay-gravel is thrown on the top of the tail race, which thus passes through a mound of earth and discharges into the open air lower down the valley. As the digging and sluicing progresses up-stream, the tail race is lengthened and the sluice boxes proper are conveyed further up the valley so as always to be near the auriferous material last uncovered. This method originated with the Chinese.

Ground Sluice.—In some cases boarded sluice boxes are not used, but a stream of water is conducted to a little trench cut in the pay dirt, which is soon enlarged by the action of the water, while the banks are at the same time shovelled or prised by the pick or crowbar into the sluice. The gold is caught in the natural riffles afforded by the uneven wearing of the bed, or rocks may be added to arrest the gold, no mercury being used. Ground sluicing is only adopted where the supply water is precarious, or the season very short, so that violent rains cause floods that would sweep away sluice boxes, and then are succeeded by dry intervals during which the boxes would warp and crack. Only the coarse gold is saved, while the duty of the water is usually much less than in wooden sluices. After a time, usually when the water gives out, the auriferous material is collected from the sluice and washed in a long-tom or cradle.

Booming.—This method of sluicing is described by Pliny in his account of gold-washing in Spain in the first century. It is adopted when the water supply is insufficient for continuous operations. A dam with a light gate, capable of being easily lifted, is built just above the part of the valley where the auriferous gravel is situated. The water trickling down the valley accumulates behind the dam, and finally overflows at one point into a small rectangular box fastened to the end of a long lever. When full of water this box depresses its end of the lever and raises the dam-gate, so that all the accumulated water rushes out at once and scours the valley bottom. As the box falls it empties itself of water, and the dam-gate returns to its original position by its own weight. This device is usually employed in connection with ground-sluicing, but a line of sluice-boxes might be used through which the sudden flood could carry gravel piled just above the head of the series.

Dry-Blowing.—If no water can be obtained, it is sometimes profitable to concentrate pay-dirt by winnowing, tossing it in a pan until the lighter particles have been blown away, and finishing with mouth-blowing. It is of course a wasteful method of concentration. In West Australia the conditions are favourable to dry-blowing, winds being strong and constant and the air hot and dry. The method used there * is to slowly empty a pan full of dirt into an empty pan placed on the ground. This operation is repeated again and again, and is followed by tossing in a pan, by "panning" as though water were being used, and finally by mouth-blowing. The larger pieces of barren material are removed by hand at intervals.

Among machines used for the purpose, the simplest consist of flat screens supported on a frame and shaken by hand, the material falling through being winnowed by the wind. In other contrivances, a bellows is added worked by the same hand-mechanism by which the screens are shaken.

In some of the machines the blast of air is used to keep the sand partly in suspension, while it is moved by gravity down an inclined table which is furnished with riffles. The auriferous material must be quite dry or perfect disintegration cannot be accomplished. In a typical dry washer, the gravel is first made to pass through shaking screens, the mesh of which is adapted to the character of the material. The object of the screen is to eliminate the larger fragments, which are usually barren. The shaking screen delivers the material on to an inclined table formed of a wire-screen, covered with light canvas or some similar material through which are forced pulsating blasts of air. These sudden puffs throw up the sand and let it settle again alternately, and as a result the light material works down the table, while the gold is retained by the riffles, being too heavy to be tossed over them by the air.

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* T. A. Rickard, *Trans. Am. Inst. Mng. Eng.*, vol. xxviii. (1898), p. 503.

In Edison's dry washer,* the auriferous sand falls vertically and is acted on by a horizontal current of air supplied by a fan, care being taken to avoid puffs and to keep the current constant. The air passes through a screen to eliminate eddies. The lighter material is deflected from its course more than the gold. As the gold is of different shapes and sizes, the gravel is carefully sized by screening. At a mill near Santa Fé, in New Mexico, where the gravel contained from 3 c. to 35 c. in gold per cubic yard, about 95 per cent. of the gold was saved. The iron was removed by a magnetic separator.†

Tail Sluices are sometimes erected to intercept the tailings from one or several sluices with the object of collecting a further percentage of gold from the waste material. These tail sluices are made of much greater size than those described above, and in some cases pay for construction. The Kumara sludge channel, erected by the New Zealand Government to carry the tailings from the sluicing works of the district into the river, caught 957 ozs. of gold in the four years ending in 1891. This sluice is 3 feet 6 inches wide, and has a grade of 1 in 28, while the boxes which discharge into it are only 18 inches to 22 inches wide. Usually, in good work, the sluices are long enough to make the tailings too poor to be worked over again at a profit, except by the Chinese, until after they have been enriched by natural concentration in the rivers.

Fly Catchers were invented in Australia for the purpose of catching the fine particles of gold, which, successfully evading the riffles of all sluices, float down on the surface of the rivers. These devices consist of weirs constructed on piles driven into the river bed, and stretching across from bank to bank of the river. Boards covered with blanketing or coarse gunny-sacking are attached to the weirs and collect all particles floating on the surface of the water. At intervals the blankets are taken up and washed in a tank. These fly catchers soon pay for their cost of construction on many rivers, but are liable to be damaged by floods, and by being used as bridges by men and animals.

River Mining.—River mining consists in the working of auriferous gravels in the channels and beds of existing rivers, and may with convenience be made to include the exploitation of deep bars below the level of the water. It is conducted in three different ways :—

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* *Eng. and Mng. Journ.*, May 9, 1903, p. 713.

† For a description of other dry or air concentrators, see Chapter ix.

1. A portion of the river bed is laid dry by damming and fluming.

2. The gravel is raised by dredgers or similar contrivances, operated from a boat.

3. Pits are sunk on the bank, and the gravels below water level excavated and raised to the surface by ordinary mining methods, or by the hydraulic elevator. The gravel won by either of these three methods is washed in the usual way by sluicing.

1. **River Mining Proper.**—This method was formerly practised on the rivers of California, but is now superseded by dredging. An entire river was frequently deflected from its course so as to lay bare a section of its bed. This was usually done by building two dams from bank to bank, with their foundations on bed rock, the water being carried off in a wooden flume, starting above the head-dam and terminating below the foot-dam.

Sometimes, as on the American and Feather Rivers, tunnels were made to drain permanently large reaches and deliver the water at a lower point. Sometimes wing-dams were built out from the bank above and below the part of the river it was desired to work, and a third dam connecting their mid-stream ends was constructed parallel to the direction of the current. The space cut off was then pumped dry.

The river bed exposed by such methods was prospected and the pay-dirt when found taken out and washed, particular attention being paid to the surface of the bed-rock. The operations were usually terminated by the autumn floods.

River mining was probably subject to more uncertainty than any other branch of gold mining. The whole capital invested was often lost, and all works and machinery swept away by a flood before the pay-dirt was sighted, while numerous instances are on record in which the alluvium on the river-bed, after having been laid bare at great expense, was not rich enough to pay for sluicing.*

2. **Dredging.**—This method of recovering gold from the gravel of river beds has of late years made remarkable progress, and its extension to the working of all flat placers has completely changed the aspect of shallow alluvial mining. A. Grothe † states that the first dredge was operated on the Clutha River in New Zealand in 1864, but even in 1891 the method had made but little progress, and was generally looked on with disfavour everywhere except in New Zealand. In dredging, gravel is raised from the bottom of the river and delivered into a barge, and the material is there washed, the gold extracted, and

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* For a full account of river mining, with details of dam-construction, &c., see the article on the subject by R. L. Dunn in the *Ninth Annual Report of the Californian State Mineralogist*, 1889, pp. 262-281. See also the *Eleventh Report*, 1892, pp. 150-153.

† *Mineral Industry for 1899*, p. 326.

the tailings sluiced back into the river. Dredges may be divid into three classes, according as (1) suction pumps, (2) continuo ladder-bucket elevators, or (3) a crane and bucket or shovel a used to raise the gravel.

(1) *Suction Pumps.*—In this system, a centrifugal pump draw material through a large suction hose reaching to the bottom o the river. As long ago as 1891, a Welman suction dredge was in successful operation at Waipapa Creek, New Zealand.* The pump was 3 feet 6 inches in diameter, and the suction pipe, which was 13 inches in diameter, could be applied at any point in an area within a radius of 40 feet. The large stones were caught and separated from the fine stuff by a riddled hopper-plate. The gold was very finely divided, and was caught on plush mats which were washed every eight hours. The water for washing was supplied from a reservoir by means of an 18-inch pipe. Stones of 56 lbs. weight were lifted by this pump, but larger ones occasionally blocked up the suction pipe.

It is difficult to regulate the relative amounts of gravel and water raised by suction pumps, the latter tending to be in great excess. The suction pipe is soon worn out, especially by coarse gravel, and the power required per ton of gravel is considerably greater than in the case of ladder-bucket dredges. Nevertheless, dredges of this type are in successful operation on the Snake River, Idaho, where a suction pump was working in 1899 on the material of the bank. A pit, 20 feet deep, was first excavated, and the sides were then sluiced down by jets of water, and sucked up through a hose pipe until a hole 200 feet square had been made, when the dredging apparatus was moved and a second pit excavated, the tailings being discharged into the first pit.† The material handled ranged from fine sand to boulders 8 inches in diameter. Suction dredges, however, are not much used.

(2) *Ladder-Bucket Dredges.*—This is, in the majority of cases, the most satisfactory and economical type, and the one in general use. Ladder-bucket dredges are made in great numbers both in New Zealand and in Western America. A typical New Zealand dredge is shown in Figs. 10 and 11. These figures are from drawings supplied to the Department of Mines of New Zealand by Mr. F. W. Payne, of Dunedin.‡ He also gave the following description of the plant.§

The hull is 119 feet long, 35 feet 6 inches wide at bow, and 50 feet wide at stern. Depth of hull forward 6 feet 6 inches, depth aft 9 feet 6 inches. The engine is 25 H.P. nominal. The ladder carrying the buckets is of sufficient length to dredge 40 feet below water line. The buckets are of 7 cubic feet

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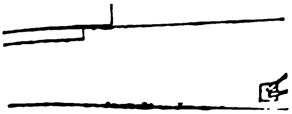
* *Reports of the Department of Mines, New Zealand.* 1891.

† *The Mineral Industry for 1899*, p. 328.

‡ *Reports of the Department of Mines, New Zealand, for 1899-1900*, p. 43.

§ [illegible]



. 10 and 11.—Typical New

capacity, and run at a speed of ten buckets per minute. The ladder is raised and lowered by a line worked by a winch, and five other winches are designed for a head line and four side lines to moor the dredge to the bank and to move it from place to place. The buckets deliver the gravel into a revolving screen 31 feet long and 4 feet 6 inches in diameter, driven by friction rollers. Thence the coarse material is directed into the main tailings elevator-buckets, and the fine material passing through the screen is delivered on to the gold-saving tables. After running over these, the tailings are deposited in a settling tank, from which they are lifted by a supplementary elevator into the main tailings elevator. The main elevator is no less than 145 feet long between centres, and is capable of stacking tailings to a height of 80 feet above water line. The capacity would be about 120 cubic yards per hour with the buckets three-fourths full. The dredge was intended to work on Fraser Flat, near Alexandria, New Zealand.

The ladder-bucket dredges in use in California bear a close resemblance to the type generally used in New Zealand, where most of the appliances originated. The following is a brief abstract of a description of a dredge placed at Oroville, on the Feather River, given by its designer, Mr. R. H. Postlethwaite:* —The hull is 80 feet long by 30 feet wide and 7 feet deep. The ladder, carrying the dredger buckets, consists of a heavy lattice girder, and the bucket belt travels on cast-steel rollers. There are 32 cast-steel buckets, each of 5 cubic feet capacity, the lips being made of nickel steel. The ladder runs at the rate of 12½ buckets per minute, giving a capacity of over 100 cubic yards per hour with the buckets three-fourths full. The gravel is delivered into a revolving screen 4½ feet in diameter and 25 feet long perforated with holes of ⅜ inch in diameter. The coarse material is conveyed away by a bucket elevator, but the fine stuff after passing over the gold-saving tables is sluiced into the river at a distance of several yards beyond the stern of the dredge. The dredge is moored by five lines and all power is supplied through electric motors. The crew consists of two men per shift, and about 70 H.P. is used in the various operations. The working costs of Risdon dredges in California are stated to be in certain cases about 4 cents per cubic yard of gravel, but are generally from 6 to 8 cents.

(3) *Crane and Bucket Dredges.*—In these dredges a single large bucket or shovel, with a capacity of 1 or 2 cubic yards, is filled with gravel and hoisted on board a barge by some form of crane. The bucket is sometimes of the Priestman grab type, opening at the bottom as it is dropped on to the gravel bed, and closing automatically when the chains begin to raise it. A large shovel is also sometimes used as in a case on the Fraser River,

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* *Mining and Scientific Press*, Dec. 15, 1900, p. 582.

British Columbia, in 1896-1897.* Here there were two barges, one containing the dredging machinery, the other the gold-washing tables. The shovel made 90 trips per hour and raised 1½ cubic yards at a time. Boulders up to 5 or 6 tons in weight could be handled with ease. The Priestman grab bucket does not close completely if a stone lodges between its jaws, and comes up partly empty, but in cases where large boulders, tree trunks, &c., are dispersed through the gravel, it might be used with advantage, instead of the ordinary ladder-bucket which cannot deal with such obstructions. The wear and tear is said to be less with a Priestman dredge than with the bucket dredge.

A Priestman dredge is shown in Figs. 12, 13, and 14, which are from drawings supplied by Messrs. Priestman Brothers.

Gold-Saving Apparatus.—The available space on a dredge is so small that long sluices are out of the question, and it is therefore necessary to disintegrate the gravel thoroughly in trommels or on shaking screens, and to make the gold-saving tables as wide as possible, so as to reduce the speed and depth of the stream of gravel running over them, especially where fine or scaly gold is abundant. The best size for the holes in the trommels is usually considered to be about ⅜ inch, but holes of 1 or 1½ inches are sometimes used. All material not passing through is regarded as worthless, and is not treated on the tables. In New Zealand the inclined tables are generally covered with coir matting or plush,† and in the United States with cocoa matting, on which is laid a sheet of expanded metal, the raised edges of which form a series of very effective riffles. On the Snake River in Idaho, the gold consists of thin flakes and is difficult to catch. The gravel is concentrated by passing through sluices lined with perforated steel plates. The fine material is then further concentrated on tables covered with burlap.‡ Similar methods have long been in use in New Zealand. J. P. Smith states § that fine gold is lost in most of the New Zealand dredges and gives a table, from which the following details are taken, of some of the most modern dredges in which fine gold is being saved :—

Dredge	Width of Tables.	Material passed over tables per foot of width per minute in cubic inches.	Proportion of sand to water.
Fraser's Flat, . . .	21 ft.	2800 cubic in.	1 to 9·43
Olrig, ...	18 ,,	768 ,,	1 to 28·8
Leviathan, ...	81 ,,	674 ,,	1 to 17·2
Waimumu, . . .	28 ,,	555 ,,	1 to 37
Mokoia and Buller Junction,	15 ,,	1152 ,,	1 to 33

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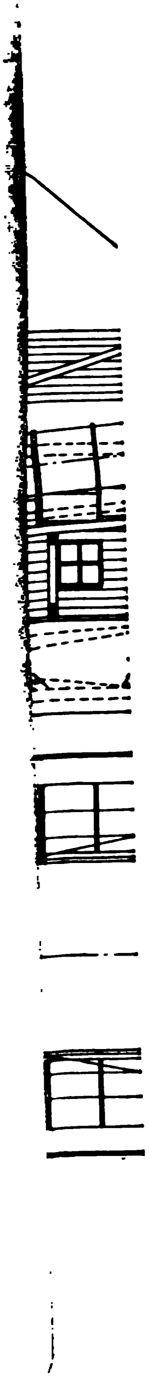
* *Mining Industry for 1899*, p. 329.

† J. P. Smith in *Report of Dept. of Mines, N. Z.*, 1899-1900, p. 51.

‡ *Mining Industry for 1899*, p. 331.

§ *Loc. cit.*



The difference between the value of ground as determined in prospecting by treating samples in a cradle and that found in actual work with a dredge shows that some fine gold is lost in practice.

The gold is recovered from the concentrates either by running them over amalgamated plates or by treatment with mercury in a clean-up pan or a rotating barrel.

Disposal of Tailings.—The tailings were formerly always discharged into the water again at some point beyond the stern of the dredge. They are now often piled on the bank to prevent any risk of mixing them with untreated gravel. Coarse and fine tailings are usually piled up separately, and when flat inland placers are being dealt with, as described below, the separated tailings occupy 50 or 60 per cent. more space than the undisturbed ground.* The bucket elevators for raising the tailings are shown in Figs. 10 to 13.

Treatment of Flat Inland Placers.—The advantages of dredges in handling gravel have appeared of late years so great that their use has been extended from the beds of rivers to all flat placer ground wherever situated. The dredge is placed in a reservoir which is filled with water. The machinery is then erected and dredging begun, the tailings being discharged at the stern, so that the reservoir and dredge move gradually forward together. By piling gravel round the reservoir and letting in more water, the dredge is raised and can be made to work its way up sloping ground, as at the Warm Springs, Idaho. According to Postlethwaite † clear water is kept flowing into the reservoir to prevent it from becoming too muddy for close gold saving.

Advantages of Dredging.—Investments in dredges have been popular in New Zealand of late years, for the reasons that results obtained have been good, the capital required is comparatively small, working expenses easily calculated and subject to few variations, and, if the claim turns out to be barren, the dredge can be transferred elsewhere or sold. The Government Inspector, in fact, reported in 1900‡ that in some parts of New Zealand no other form of mining would be entertained as an investment. Dredging on rivers is a far cheaper method of treating the gravel than any other, but the great extension of the future will doubtless be in inland work. The amount of water required is small, the risk of loss by floods or any kind of accident infinitesimal, and the labour costs trifling.

Difficulties in Dredging.—The difficulties of raising the gravel due to the occurrence of tree trunks or large boulders can be overcome now that vacuum pumps have been superseded. The chief difficulty now is to save the fine gold in the gravel. The losses of gold are often large, though seldom considered. In one New Zealand dredge, it was found as the result of treating measured quantities of tailings that gold amounting to 2 grains per cubic yard of tailings treated was lost.[*] The yield on the dredge was about 3 grains per cubic yard, so that only 60 per cent. of the gold was saved. When there is an "overburden" or layer of clay above the auriferous material, it is found that lumps of clay and water carrying clayey matter in suspension are responsible for the losses of gold. The best method of avoiding these losses is to strip off the overburden of clay before attempting to raise and wash the auriferous gravel.

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* *Mineral Industry for 1899*, p. 331.

† *Mining Mag.*, Jan., 1905, p. 8.

‡ *Report on Dept. of Mines of N.Z.*, 1899-1900, p. 41.

The bottom on which the gold-bearing material rests is also of importance. A hard bed-rock prevents the gold lodged in its crevices from being removed by the dredge. Soft bed-rock partly decomposed *in situ* is, on the other hand, an excellent bottom for dredging. A soft tenacious clay underlying the gravel causes loss of gold in washing.

Cost of Dredging.—The capital expenditure on a dredging plant in New Zealand varies from about £2,500 for a small dredge, with a theoretical lifting capacity of 60 cubic yards per hour, to £10,000 for a large dredge capable of lifting 150 cubic yards per hour. The working cost of a small dredge is given as about 1 ounce of gold per day in New Zealand,[†] or about twopence per cubic yard of gravel treated. The cost of working in Western America is stated to be about four cents per cubic yard in a case where the power was supplied by steam from coal.[‡] Working costs at Oroville, California, are given by Postlethwaite as ranging from 3·66 to 8·7 cents per cubic yard.[§] Much larger dredges, handling 7,000 cubic yards or more per diem, will soon be in operation in California, and the cost on these may be less. The working cost on a dredge of moderate size is given by F. W. Taylor[‖] as about £40 per week, and the return may be expected to be from 20 to 30 ozs. of gold per week, so that the profits are often large. A great future for dredging appears to be promised in Siberia, where the industry was started in 1900. The working cost was given by Taylor as about 4·2d per cubic yard in 1904.

Method of Working Siberian Placers.—The methods and apparatus employed in Siberia differ so markedly from those which have been adopted elsewhere, that they are well worth a special description, although they cannot usually be applied to placers found in other parts of the world, owing to the difference in economic conditions. In California the valleys are narrow and the grade steep, so that watercourses are usually close to the auriferous deposits, and the sluices can be made of almost any length, while still conforming to the general slope of the soil. In Siberia the slope of the valleys is so gradual that the flow of the water is almost imperceptible, and takes place through wide marshy tracts. The result of this is that the sluices must be short, being usually less than 100 feet long, and their upper ends must be raised on trestles. As a further consequence, also, the gravel must be excavated by hand and carried in waggons to the sluice, and the tailings removed in a similar way, the flow of water acting by gravity not being available for these purposes. The excavation is made in benches or terraces, working up the valley. The height of each bench above the lower one is about 5 feet, and the gravel is picked down from the face of each bank and shovelled into carts by which it is conveyed to the washing establishments. The additional expense entailed by these causes is balanced by the low cost of labour in the country, and an incidental advantage lies in the fact that the washing apparatus can be placed outside the limits of the river during flood time, and so may remain for a number of years undisturbed, while all the gravel in the district is being washed. Only surface deposits are exploited, no deep workings existing in the country.

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[*] *Report on Dept. of Mines of N.Z.*, 1899-1900, p. 41.

[†] *Loc. cit.*

[‡] *Mineral Industry for 1899*, p. 336.

[§] *Mining Mag.*, Jan., 1905, p. 10.

[‖] *Engineering and Mining Journ.*, Jan. 14, 1904, p. 82.

There are three types of apparatus employed, each designed for washing a particular kind of deposit. They are:—

1. The Siberian sluice, which is used to wash light sand.
2. The Trommel, used for loamy sands.
3. The Pan, used for gravel which is cemented together by means of compact clay.

1. *The Siberian Sluice.*—The apparatus at Voltchanka, which may be taken as a type, consists of a head sluice and three secondary sluices, which are placed at right angles to the head sluice, and which leave it at different points and converge to a common centre, where the tailings are discharged. The head sluice begins at a height of 13 feet from the ground; it is about 90 feet long by 2 feet wide, and has a fall of about 1 in 14. The sands are dumped from the waggons on to a wooden platform situated above the sluice-head, and shovelled into the latter, a stream of water being turned in at the same time. After passing through a grizzly, the gravel runs over a series of cast-iron cross-bar riffles, which form a number of rectangular depressions (or *pigeon-holes*) in the bed of the sluice, by which the disintegration is favoured. The stream then flows over an iron screen, through which a part of it falls into the first secondary sluice, while the remainder continues its course over more pigeon-hole riffles. This arrangement resembles the Californian undercurrent. A second and a third screen open on to the other secondary sluices, and the part of the gravel (consisting chiefly of small stones) which has resisted disintegration, and has not passed

through the screens, is then let fall into a hopper, whence it is removed to the tailings waggons.

The secondary sluices are wider than the head sluice, and have a steeper grade, and the amount of water and auriferous material passed is of course much less in each one than on the principal sluice. The sands first pass over a number of transverse riffles, and then over about 30 feet of blanketing, the sluice being widened at the same time, and subdivided by longitudinal wooden partitions, and the grade being as much as 1 in 6, while small drops are introduced at intervals of a few feet. The third sluice has a more gentle inclination than the others. The tailings fall into a shallow sump, and are instantly lifted out of it by a bucket elevator or "tailings-wheel" operated by water power, and stored in a hopper, whence they fall into waggons, by which they are removed to the dumping ground.

No mercury is used in these sluices, and only the production of "grey concentrates" is attempted, this work being continued from 6 a.m. to 7.30 p.m. every day, after which the concentrates are collected from the riffles. They consist of gold in scales and plates, magnetic iron oxide, pyrites, rutile, together with some quartz, &c. They are treated with mercury in the Siberian trough or on inclined tables, the method being that described on p. 51.

The apparatus described above treats 500 tons of gravel per day, the labour required being furnished by twenty men and ten horses. The gravel treated contains an average of from 12 to 15 grains of gold per ton, rarely falling below 6 grains per ton; the exceptional richness of 3½ dwts. per ton has been observed. The gold is chiefly found in the head sluice, where 70 per cent. is retained, 30 per cent. being caught on the secondary sluices. At a similar establishment at Tchernaia-Retchka, however, where the gold is less finely divided, 97 per cent. was caught on the head sluice, and only 3 per cent. on the secondary sluices. The amount of water used at Voltchanka is about six times the weight of the gravel. The cost of construction of the works was 70,000 roubles, or about £7,000.

2. *The Trommel.*—Gravels which are too compact for satisfactory disintegration in the short sluices described above are subjected to a preliminary treatment by a trommel. At Berezovsk the trommel is of sheet iron of 9 mm. thick, having holes in it of about 1 mm. in diameter. The trommel is about 12 feet long, 3½ feet in diameter at one end, and 4½ feet at the other, and is set inside with denticulated plates of iron to assist in the disintegration effected by the water. The machine is driven by a water wheel, and is sufficient for the disintegration of from 400 to 500 tons of gravel per day, requiring the expenditure of about 3 horse-power to drive it. The amount of water used in the trommel and on the tables is 67·5 litres per

second, or about seven and a-half times the weight of the ore. The washing is effected on inclined tables only 30 feet long and 12 feet wide, and with a grade of about 1 in 4, placed with the incline at right angles to the length of the trommel. Near the head of the table, and stretching across it, is a deep trough-like depression, and below this there are a number of transverse riffles, in which grey concentrates are caught and treated as usual. The Berezovsk establishment employs twenty-five men and fourteen horses constantly; the trommel usually lasts for two seasons.

3. *Pan Washings.*—Sandy clays cannot be economically disintegrated in a trommel, and are, therefore, treated in a washing pan, which bears a strong resemblance to the cement pans employed in California. The pan usually consists of cast iron, and is from 8 to 16 feet in diameter, with vertical sides from 1 to 5 feet high. The bottom is of cast iron or sheet iron, and has numerous holes in it of about $\frac{1}{16}$ inch in diameter, widening downwards. The bottom is divided into 25 sectors, between which are deep grooves for the collection of the pebbles. Through a circular opening in the centre of the pan there passes a revolving axis to which are suspended eight horizontal arms studded with vertical iron teeth, some of these being shaped like plough-shares. The revolution of these arms effects the disintegration of the sandy clays, which are fed into the pan together with water and puddled until fine enough to pass through the holes in the bottom, and the stones are removed at intervals by opening little gates placed opposite the radial grooves. The disintegrated gravel falls from the pan on to concentration tables, similar to those used after disintegration in the trommel. At Berezovsk the pan is $11\frac{1}{2}$ feet in diameter and 5 feet deep, and the arms revolve at the rate of 25 turns per minute. From 50 to 55 tons of material are treated in twelve hours, the water consumed, including that required for power, being ten times the volume of the sand.

Beach Mining.—Beach mining is a comparatively unimportant form of shallow placer mining. The sea beaches on parts of the coasts of California, Australia and New Zealand contain small quantities of gold, which have been proved, in all cases in which the matter has been investigated, to be derived from the cliffs, in which the gold is generally less concentrated. Some streaks of black sand, however, in the "Gold Bluff," California, have yielded $135 or $6\frac{3}{4}$ ozs. per ton by actual working.* The waves of the sea wash down and partially concentrate the poor sands, and, under certain rather exceptional circumstances, as the tide goes out the surface of the beach is left covered with black sand, in which numerous specks of gold occur. This is carefully scraped up and transported inland to be washed, as sea water is not well adapted for the purpose, although it is used by one Californian company. The next tide usually washes away all the valuable material which has not been collected, or else covers it with barren sand. There is great difficulty in washing the black sand in California, as it consists largely of rounded grains of magnetite, the density of which is about 5·0, while the gold is in minute flakes and scales, which can be seen under the microscope to be oblong in shape, and thicker at the sides than in the middle (a shape due to continued pounding of a malleable material). This form is so easily moved and buoyed up by water that it is difficult to get a "colour" with the pan, and the amount caught by the mercury in sluices or long-toms is usually an insignificant proportion of the total assay value of the sand. The industry is generally a languishing one, but the discoveries of rich deposits on the beach at Cape Nome, Alaska, resulted in great activity. In 1899, rockers of all sorts and descriptions were employed, and in 1900 when sluices and machines of various designs were used the production rose to £1,000,000.

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* *Prod. Prec. Met., U.S.A.*, 1884, p. 557.

Treatment of Shallow Placer Gravels by Steam Shovels and Amalgamated Plates.—A plant for the treatment of placer deposits, which are similar in nature to those worked in Siberia, has been devised in America, where labour-saving contrivances are indispensable in all such cases, owing to the high rate of wages. The apparatus consists essentially of a combination of a steam shovel or navvy, and an amalgamator consisting of a wide waggon-shaped wrought-iron trough loosely lined with silver-plated amalgamated copper plates, which form a series of steps or riffles at the sides of the trough. The material is elevated into a hopper by the excavator, and thence is charged into a revolving trommel placed inside the trough. Here the disintegration of the gravel is effected, and the fine material falls through into the trough, while the stones are discharged outside at the end of the trommel. At the bottom of the trough there is a water pipe, carrying water at high pressure, and in that pipe a series of jets pointing alternately forwards and backwards. The result is to give a series of eddies or whirlpools in the water in the trough, and the sand and fine gold is continually carried up to the surface near the middle line of the trough, and in descending again near the sides it comes in successive contact with several of the plates, which form a series of steps. The fine sand is eventually discharged at the end of the trough. It is stated by the Bucyrus Steam Shovel and Dredge Company, by which the machinery is manufactured, that such a machine erected in Montana has a capacity of from 600 to 800 cubic yards of gravel per day, all the water required being supplied by a pump raising 500 gallons per minute, the proportion being only three of water to one of gravel, which seems much too small. It is stated that gravel containing only 12 cents of gold (*i.e.*, about 3 grains) to the ton has been treated successfully by this machine, but exact records of continuous work done by it are wanting. It will be noted that special means must be adopted in each case for the handling of the tailings.

In 1903, a steam shovel plant was at work on Dominion Creek, Klondike.* The bucket shovel had a capacity of about 0·75 cubic yard, and excavated 800 cubic yards in a day of 20 hours. The ground could be cleared to a depth of 10 feet by this plant. The gravel was emptied into a hopper and sluiced, thence to a trommel from which it passed to sluice boxes. The value of the gravel was $1·50 per cubic yard and the working costs were 13 cents per cubic yard.

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* E. Haanel, *Eng. and Mng. Journ.*, April 11, 1903, p. 559.

CHAPTER V.

DEEP PLACER DEPOSITS.

Nature and Mode of Origin of Deposits.—Both in Australia and California, besides the superficial placer deposits situated in or near the existing rivers, which in the deep cañons of the Klamath and other rivers in the extreme north of California attain a thickness of 250 feet, there exist auriferous gravels which bear no apparent relation to the present drainage of the country. These gravels often attain enormous thicknesses, and are in many places covered by volcanic rocks, consisting of basaltic lavas and tuffs, which are sometimes interbedded with gravel and loam. This latter circumstance shows that intermittent action of the volcanic vents, with long intervals of repose, has taken place. There has been some difficulty in accounting for the origin of these deep placers, and it has been ascribed in succession to the agency of the sea, of ice, and (for California) of a huge river flowing from north to south at right angles to the direction of flow of the existing rivers. None of these views are now entertained, and the "fluviatile" theory is generally accepted, the origin of the gravel being ascribed to the depositions of ancient rivers flowing in courses roughly parallel to those of existing rivers. The geological age of these ancient rivers has not yet been determined with certainty, but though they may be Pleistocene, the balance of palæontological evidence is perhaps in favour of Whitney's view that the deposits were formed in the Pliocene period.

The ancient Californian rivers probably had their sources at somewhat higher altitudes than those now existing, and had more uniform general grades, the slope of their beds corresponding more nearly to the general slope of the country. The existing rivers, on the other hand, have steep grades in the upper parts of their courses, followed by comparatively level stretches below. The old rivers, however, like their successors, had rapids, falls, and level stretches, the grade varying from 5 feet to 250 feet or more per mile. The Pliocene rivers ran in valleys which were broad and shallow in comparison with the present deep precipitous cañons, and the volume of water was in general much greater than that delivered by their representatives of to-day. The width of the valleys varied from 100 feet to fully 1½ miles (which is the width at Columbia Hill), and the depth must have been often over 1,000 feet. These valleys were already partly filled up by accumulations of gravel, when the outbreak of volcanic activity in many cases filled up the remainder, and the streams were deflected into other channels, which often lie close alongside the old cañons. These new channels have been excavated by the running water until they now lie much below the level of the beds of the Pliocene rivers, and consequently the gravels which were deposited in the old valleys now sometimes crown the highest ground in the district, the general level of the country having been greatly reduced in height. The new channels have been cut partly in the old country rock and partly in the Pliocene auriferous gravels and their covering of volcanic rocks. Sometimes the course of the present cañons cuts that of the old at several points owing to the sinuosity of both, see Fig. 15, in which A represents the modern river, and B the ancient one. The result is that sections of the old valley from bed-rock to surface are exposed in the sides of the cañons, usually at some height above the present level of the water, and it was at such points as these that the discovery of the existence of the deep placers was first made. The hard covering of basalt has served to protect the more friable gravels, which have been for the most part removed in those places where the lava has been worn away or has never existed, so that the largest tracks of gravels still existent lie beneath the volcanic rocks.

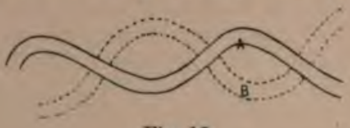

Fig. 15.

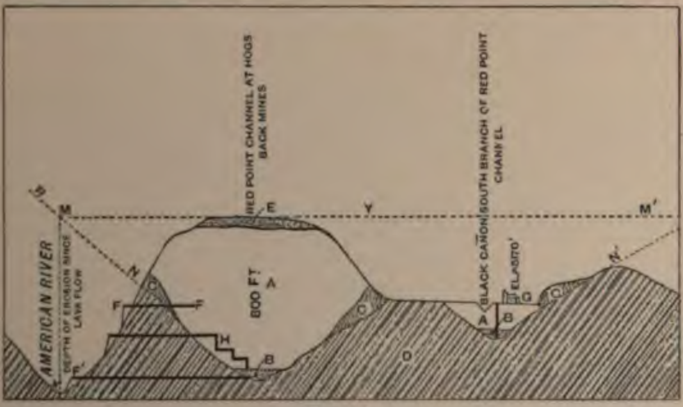
AMERICAN RIVER
DEPTH OF EROSION SINCE LAVA FLOW
RED POINT CHANNEL AT HOGS BACK MINES
BLACK CAÑON
SOUTH BRANCH OF RED POINT CHANNEL
ELASITO'
800 FT.
M
E
Y
M'
N
N'
A
B
C
D
F
F'
G
H

Fig. 16.

Fig. 16 represents a section across two ancient channels (B, B) and a modern cañon, that of the American river. Here, A is the volcanic capping, which is 800 feet thick above the Red Point channel; B, B are the auriferous gravel channels; C, C are deposits of gravel on the "rims," containing gold in places; D is the bed-rock, consisting of dark-blue slates; E is a barren deposit of angular débris and boulders; F F are prospecting tunnels, which were put in at too high an altitude; F′ is the tunnel bored with the object of reaching the bottom of the gravel deposit; H are prospecting winzes sunk in order to discover the position of the gravel. The space included within the dotted lines N M Y M′N′ has been obviously denuded since the deposition of the volcanic cappings, the soft slate rims N M R and N′M′ having been worn away, while the hard lava has resisted erosion. The vertical depth from M to the American river is about 1,800 or 2,000 feet.

This condition of things is that prevailing in California, but in Victoria the structure closely resembles that just described, with the exceptions that the old valleys were smaller and that the erosive action of the rivers since the deposition of the basalt has been comparatively slight, owing to the slight grades of the streams caused by the low elevation of the country and to the small amount of the rainfall. In consequence of this the basalt has usually not been worn through, and the "deep leads" or old river bottoms are often below the level of the present streams, so that although a larger proportion of the Pliocene gravel remains, it is more difficult and expensive to mine.

The shallow placers, at any rate in California, have resulted in the main from the erosion of these deep placers, the materials of which, having undergone a natural concentration in the ground sluices afforded by the river beds, furnished the wonderfully rich river-bed and bar deposits, which yielded so much gold between 1848 and 1860. The deep level gravels consist in slaty districts chiefly of quartzose sand, the fine materials furnished by the disintegration of the slate having been for the most part swept away, and the products of the quartz veins contained in the slate being left. These are the only gravels which pay for treatment. In granite districts, where the gravels are composed of more heterogeneous materials, and in cases where they consist of volcanic boulders and detritus, little or no gold is found. The lower parts of the gravels are often cemented into a conglomerate, called "cement," by infiltration of silica, oxides or sulphides of iron, or, rarely, carbonate of lime; when the gravels are covered with lava, the whole thickness is in some cases converted into cement.

Distribution of Gold in the Gravels.—The gold is found chiefly either in contact with or just above bed-rock. If this consists of soft slate, and especially if the planes of cleavage are at a high angle to the horizon, particles of gold are often found in the natural riffles thus formed, and are disseminated through the rock to the depth of a foot or two. If depressions, pot-holes, or fissures exist in the old river bottom, they are usually very rich in gold. Where, as often happens, there is a channel, or "gutter," to adopt the Australian expression, cut by the stream in the lowest part of the valley, the gravel filling it is usually much richer than that found elsewhere. Such rich portions, often only a few feet wide, and of insignificant depth, but extending to considerable distances in the direction of the stream, are called "leads." Rich streaks also occur at various levels in the gravels, often resting on "false-bottoms," which consist of impermeable beds of clay or some similar material. Sometimes these streaks are richer than those encountered at bed-rock, as, for example, at the Paragon Mine, Placer County, California.

The amount and position of the gold vary, as in the case of the present rivers, with the grade, the shape of the valley, the volume of water, the amount of gravel being carried down, &c. "An underloaded current—*i.e.*, a current charged with less detritus than it is well able to carry—is apt to cut its bed, and prevent the accumulation of gravel. A greatly overloaded current will deposit too rapidly to admit of the concentration of the gold dust."* Under conditions intermediate between these extreme states, the current may be just strong enough to keep its bed clear from all accumulations except a small quantity of coarse gravel and the coarse gold, which is caught in the natural riffles, and thus all the conditions necessary to form a rich bed of pay-dirt may be present. If, however, the bed consist of granite or other rock which wears in smooth and rounded shapes, little gold will be caught. Slates, consisting of layers of uneven hardness, wear irregularly, and afford a good gold catching surface. The conditions noted above as necessary to form rich gravels cannot be expected to have been prevalent over great distances. "An increase of grade or narrowing of the channel will cause an increase of velocity, and the same stream may be underloaded in a narrow steep section, and overloaded in a broad flat section."† The difference of velocity between the middle and sides of a stream, and between the inside and outside of a bend, may give the right conditions in one part of a river bed and not in another. Thus with high grades, rich gravels should occur in the less rapid, and with low grades, in the more rapid parts of a stream. Having regard to such considerations, the richer parts of existing rivers can be pointed out with little trouble. When, as in the Pliocene rivers, the beds are buried to a depth of hundreds of feet, the richer parts are more difficult to find.

The history of the Pliocene rivers began with a period when excavation exceeded deposition, when the rivers were underloaded for at least a portion of each year, and the channel was constantly being deepened. Some bench or terrace gravels were formed at this time, and being at the sides of an underloaded river tended to be rich. The river bed, although rocky and comparatively free from sand, would perhaps accumulate some coarse gold, which as the channel deepened was no doubt in part ground up into fine particles and carried off, but at the time when the excavation had reached its lowest point, some of this coarse gold would certainly be present. When the underloading of the stream ceased, whatever caused the cessation, a pause must in many cases have occurred before the gravel proved too much for the stream to carry. During this pause the conditions for gold catching were favourable, and hence rich gravels were formed on bed-rock in the gutters or channels. Then, as the streams became overloaded, sand and gravel accumulated rapidly, so that little concentration of the gold in them could take place. The rivers flowed over thick sand banks and, in consequence, frequently changed their courses. The sands, being deposited by overloaded rivers, of course contained fewer and smaller boulders, and the thick masses of poor sand thus went on accumulating until the volcanic outbursts put an end to the process.

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* Ross E. Browne, *Tenth Report of the Cal. State Mineralogist*, p. 448.

† *Ibid.*

Origin of the Gold in the Placers.—The origin of the gold in deep placers has long been a vexed question. It was formerly accepted without question that the erosion of auriferous quartz lodes existing at higher altitudes furnished both gravel and gold. In support of this it was urged that the same districts which furnished auriferous gravels abounded in quartz veins at higher levels, while Whitney pointed out that numerous lodes were intersected by the valleys and were still to be seen in the bed-rock. On the other hand, the fact that the nuggets found in the drift are much larger than any masses of gold encountered in veins, and that the placer gold is of superior fineness, are difficulties in the way of accepting this theory. Moreover, Egleston states that nuggets as large as a man's fist have been found embedded in the midst of fine sand, whither they could not have been carried by the action of running water, but authentic instances of such finds seem to be lacking, nuggets usually occurring in coarse gravel, among boulders. It is further declared by the opponents of the erosion theory that if a small quantity of soft material like gold mixed with lumps of hard quartz, were washed down by water, then, long before the quartz, could be reduced by grinding to the condition of grains of sand, the gold would be worn down to such a fine state of division that none of it could lodge in the river bed at all. In opposition to this contention, it may be urged that the extreme malleability of fine gold would make this comminution very slow, and scales of the metal are said to have their edges blunted and thickened by the pounding action of dry sand moved by the wind, instead of having them worn away. Moreover nuggets often include more or less quartz.

In 1864, in order to account for these and other facts, Mr. A. C. Selwyn, of Victoria, suggested a theory of solution in which it is supposed that the gold disseminated through the rocks and drifts is dissolved by percolating waters which contain acids and salts in solution, and is reprecipitated around certain centres. Selwyn considered that the waters capable of dissolving gold must have acquired this property by passing through the beds of basalt, &c., overlying the drifts, inasmuch as large nuggets occur, in districts where basaltic eruptions have taken place, while, where these are absent, the gold is very fine, and nuggets can scarcely be said to exist. The fact has long been known that gold is soluble in certain dilute solutions of salts, likely to be met with in nature, such as a mixture of nitrates with chlorides, bromides or iodides, or as the haloid ferric salts. This has been firmly established by the researches of Skey, Daintree, Egleston and others. Also, the precipitation of gold from these solutions around nuclei consisting of particles of gold, pyrites, &c., by organic matter present in the liquid, has been studied, and efforts made to form nuggets similar to those found in nature, without much success. This, however, is not surprising since the conditions in nature, including almost unlimited time and immense quantities of exceedingly dilute solutions, cannot be reproduced in the laboratory. Among other pieces of evidence against the erosion theory which have been cited, may be mentioned the fact that some gold placers occur at higher levels than any quartz veins yet discovered or likely to be discovered; also that nuggets are said to have been found embedded in decomposed rocks in positions to which they could not possibly have been carried by running water, so that these nuggets at least must have been formed by accretion. The prevalent belief among diggers that the tailings from sluicing operations grow in richness so as to be worth working over again after a few years is explicable on either hypothesis.

The exponents of the erosion theory have pointed out that the fineness of the placer gold may be accounted for by supposing that the impurities (silver, copper, &c.) formerly present in the native gold have been dissolved away by meteoric water, in which they are much more soluble than gold is. The existence of large masses of gold in placer deposits was accounted for by Whitney by assuming that the upper portion of the lodes, now washed away, were richer, and contained larger masses of gold than the remains of the lodes now left, but Liversedge has shown* that this assumption is not necessary. Some nuggets too have been found showing undoubted signs of erosion by water, but these are rare. Liversedge has also adduced evidence (*loc. cit.*) that, even if the small particles of gold found in placers have grown by accretion, nuggets cannot have appreciably increased in size. The suggestions made to account for the great richness at bed-rock—viz., that gold has "settled" through the quicksands, or that the gold solution has remained longest in contact with the sand nearest bed-rock—are not wholly satisfactory, and must be supplemented by some such explanation as that given above, p. 74.

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* *Proc. of the Royal Soc. of N.S. Wales*, Sept., 1893.

The view is now generally accepted that placer gold has almost invariably resulted from the erosion of older auriferous deposits.

The gold in the gravels of the Klondike region differs from ordinary placer gold. It is often in large nuggets, including much quartz, and is usually rough and but little water-worn.* It is of low standard, and has resulted from the erosion of auriferous rocks by glaciers which have carried the broken material into the valleys and left them as moraines.

Materials occurring in the Placer Deposits.—In California, if quartz grains and silicified wood are excepted, the most abundant mineral is black iron-sand, which usually consists of magnetite, although menaccanite, a form of hematite in which part of the iron is replaced by titanium, also occurs. These minerals must have been derived from the lavas, as neither of them are known to occur in the quartz veins of the country. Platinum and its allies are usually present in more or less abundance; thus iridosmine occurred to the extent of 1 in 100,000 of the gold in the early days, and increased afterwards to 15 or 16 times that proportion. It is still abundant in the beach sands of Northern California. Grains of native copper, nickel, and perhaps lead have also been detected, and a few diamonds occur, while garnets, small crystals of zircon and cinnabar are very abundant.

METHODS OF TREATING DEEP PLACER GRAVELS.

Both in California and Australia, when the first gold discoveries were made, the river beds and bars were at once explored, and soon afterwards the flats closely adjoining them. Subsequently, the bench gravels situated in the same valleys, and the side ravines and gulches, which remained dry during most of the year, were prospected and worked, owing to the rapid growth of the mining population and to the fact that the exhaustion of the shallow placers was already beginning to make itself felt. The result was that the exposed edges of the outcrop of some of the deep leads were found, and the pay-dirt followed into the hill-side by drifting. Then, as in many cases it was found that the gravels overlying the pay-dirt on hill-sides, although poor by comparison with the earth below them, nevertheless contained a small quantity of gold, the idea was evolved of breaking down the whole bank by jets of water, and passing all the material through the sluices. Hence arose the practices of drift and hydraulic mining,* of which the former is largely used in California, while the latter was much used in California prior to 1884 and in New Zealand and throughout the Western States of America, but cannot be applied in Australia or Siberia owing to the general flatness of the country. In Siberia, only shallow placer deposits are worked. In Australia, the deep leads are usually reached by shafts, since the surface of the country is not intersected by deep cañons, as in Western America. Drift† and shaft mining of deep leads will not be further referred to in this volume, attention being confined to the treatment of the gravel after it has been raised to the surface. In hydraulic mining, the breaking down of the gravel is so intimately connected with the extraction of the gold that a short description of the whole process is given.

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* J. B. Tyrrell, *Amer. Geol. Soc.*, Dec., 1898.

Hydraulic Mining.—This method of working consists, as has been already stated, in breaking down banks of gravel by the impact of powerful jets of water, and passing the disintegrated material through a line of sluices, without the agency of hand labour. The chief requisites for the successful application of hydraulic mining are—

1. Large quantities of auriferous gravel, not less than 30 feet in thickness, and not overlaid by any appreciable thickness of barren material, which would necessarily be passed through the sluices with the pay-dirt. The gravel treated need not be rich, a mean yield of less than 1 grain of gold per cubic yard being often enough to furnish profits if the operations are on a sufficiently large scale.

2. A plentiful and uninterrupted supply of water throughout considerable portions of the year.

3. Sufficient fall in the ground so that (*a*) the water may be delivered under the pressure of a head of from 100 to 300 feet, (*b*) the tailings can be easily carried away to a large dumping ground, which is most conveniently, either the sea or a large and rapid river.

Commencement of Operations.—In California, the naturally occurring banks or cliffs in the gravels in the sides of the gulches were first selected for attack. Later, some of the deposits occurring in those channels which are not intersected at favourable points by the present system of drainage were operated on. It is necessary in such cases to run a tunnel from the nearest cañon in the bed-rock to the lowest point in the gravel, this point being found or guessed at by prospecting operations. The tunnels are often of great length, that at the North Bloomfield Mine, Nevada Co., Cal., for example, being 7,874 feet or 1½ miles long. One or more shafts are then sunk from the surface through the gravel to the tunnel, and washing operations are begun by ground-sluicing, letting the water and gravel fall down the shaft and run through the tunnel, in which the sluices are sometimes laid. The surface near the shaft is thus gradually lowered, or it may be terraced by hand labour, until an excavation is made of sufficient size to enable the ground to be attacked with the hose. Washing then proceeds regularly in this manner, the bank being broken down by jets of water, and the products being allowed to fall down the shaft and pass through the tunnel.

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* The invention of hydraulic mining is ascribed to Edward Mattison, of Sterling, Connecticut, who used the method in 1852 on a small scale. For all details in connection with the industry, see *Hydraulic Mining in California*, by A. J. Bowie, New York, 1885. This is still the standard work on the subject, and has been largely used in drawing up the summary which follows.

† A detailed description of Drift Mining is given by Russell L. Dunn in *Eighth Report of Cal. State Min.*, 1888, p. 736.

The Supply of Water.—The amount of water required by large undertakings is far more than can be obtained from the rainfall on the hills immediately round the mine. Thus the North Bloomfield Mine, in the season of 1877-8, used between sixty and seventy millions of gallons per day, or enough for the total supply of a city of three million inhabitants. The workings, moreover, must necessarily be considerably above the level of any large rivers in the neighbourhood, so that it is often necessary to construct huge reservoirs at convenient spots to store up the rainfall and melted snows of large districts, and to convey the water thence to the mine by ditches, flumes, or pipes. Wooden flumes are cheaper than ditches, but are more liable to be damaged. Sheet-iron pipes are made as much as 30 inches in diameter, and are cheap, durable, and easily repaired; they entirely prevent losses by leakage and evaporation, and deliver more water under similar circumstances than flumes, as they offer less frictional resistance to the flow.

Where it is necessary to cross side cañons or gulches the flume or iron pipe is carried over on a light trestle bridge, or the water is passed through an inverted siphon formed by a wrought-iron pipe which passes down one side of the valley and up the other, being filled from a head-box or reservoir, and delivering the water at a lower level on the other side. At Cherokee, Butte County, an inverted siphon pipe is used to carry the water across a ravine 873 feet deep. The diameter of the pipe is from 30 to 34 inches, and its greatest thickness (where there is a pressure of 384 lbs. per square inch) is 0·375 inch.* The Miocene Ditch Company, operating in the same County, carried their flume, of 4 feet wide and 3 feet deep, round the face of a bluff 350 feet high, supported on L-shaped iron brackets made of bent T rails, soldered into holes previously drilled by men let down the face of the cliff by ropes. Successful ditch construction often depends on such engineering feats.

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* *Ninth Report Cal. State Min.*, 1889, p. 125.

The water is delivered at a convenient height above the workings into a "pressure-box," consisting of a small wooden reservoir from which the pipes take their origin. In many cases the reservoirs and ditches are owned by separate companies, or, as in New Zealand, by Government, and the water is sold to the miners by measure. The unit in New Zealand is a "government head," and in the United States a "miner's inch," following the system in vogue in Spain and Italy. The amount of water that will flow through an orifice 1 inch square, cut in a board of 1 inch thick, under a head of water that varies with the custom of the locality, but is usually from 4 to 8 inches, is called a miner's inch. The amount of flow in twenty-four hours is called a "twenty-four-hour inch," and similarly there are ten-hour and twelve-hour inches. The quantity of water in a miner's inch varies with the head of water used and the form and size of the orifice for delivery. Thus the amount delivered from an orifice 25 inches long and 2 inches wide is reckoned as 50 inches, although it will be more than fifty times as much as the delivery from an orifice 1 inch square. The twenty-four-hour inch under a head of 7 inches amounts to about 2,230 cubic feet.*

The water is conveyed from the pressure-box by pipes, which were formerly made of canvas hose, to which iron rings 3 inches apart were added for pressures of over 100-feet head. They are now made of sheet iron. The iron feed-pipes are made from 10 to 15 inches in diameter, and the thickness varies according to the pressure which they may be called on to withstand. Sharp bends in them are avoided, as the flow of water is checked thereby. They are liable to collapse if the level of the water in them is reduced, and a partial vacuum formed inside; hence, as in the case of all other sheet-iron pipes used in hydraulic mining, they are fitted with valves, which are constructed so as to freely admit air from without. The water is discharged through a nozzle called a "giant" or "monitor" (Fig. 17). The nozzle was at first a sheet-iron tube, having an aperture 1 inch in diameter, and was held in the hand. The size of the nozzle was gradually increased, until it reached a diameter of 11 inches. Such a stream, under a head of 200 feet, requires special appliances to control it, deflect it at will, and prevent the nozzle from "bucking."

Breaking Down the Bank.—When the jet is first directed against the bank, the water spatters in all directions, then buries itself a little, and after a time, in loose ground, a "cave" takes place, the undermined bank falling down. By the method of undermining, the power of the giant is much increased, especially where hard and soft layers alternate. When large caves are about to take place the water is turned off, as otherwise the ground may run so far as to overwhelm the monitor and the workman directing it. The nozzle is placed as near to the bank as possible, consistent with the safety of the workers, so as not to waste too much of the initial velocity of the stream of water. Consequently, lofty banks are not advantageous, and if they exceed 200 feet they are usually worked in terraces of 100 feet or so in height. In some parts of the Spring Valley Mine, however (see Fig. 18), a bank of 450 feet high was worked in a single bench, and it was then not unusual for the runs of ground to bury pipes which were throwing 7-inch streams from a distance of 400 feet from the face of the bank.

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* For further details concerning miner's inch *vide* Art. by P. M. Randall in *Precious Metals in the United States*, 1884, pp. 558-572.

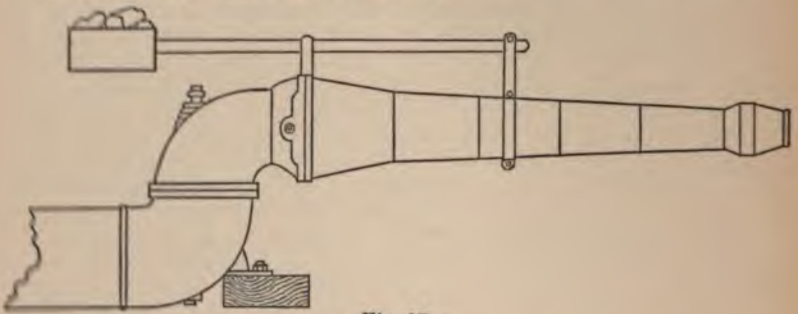

Fig. 17.

The jet is, if possible, delivered unbroken against the face of the bank, as its disintegrating power is thus kept at its maximum. However, in some cases, where it is cemented, the gravel is too hard to be economically broken down by the water alone, and blasting is then resorted to, a drift being run into the bank, and cross-cuts made at the end in which the powder is placed; the drift is then filled up, and the charge exploded by electricity. It is more economical to blow out the base of the bank, as the upper part then falls by its own weight and can be broken up by the water. Sometimes arrangements are made to explode very large blasts: thus, at the Blue Point Mine, Nevada County, a charge of 50,000 lbs. of powder was exploded in cross drifts at the end of a main drift 325 feet long in the year 1870, and 80,000 cubic yards of gravel were brought down, while at another mine, 3,500 lbs. of dynamite were exploded in 1872, and 200,000 cubic yards of gravel thrown down.

Washing the Gravel in the Sluices.—The sluices in which the gold is caught are constructed on exactly the same principles as those already described, but are larger and, though usually

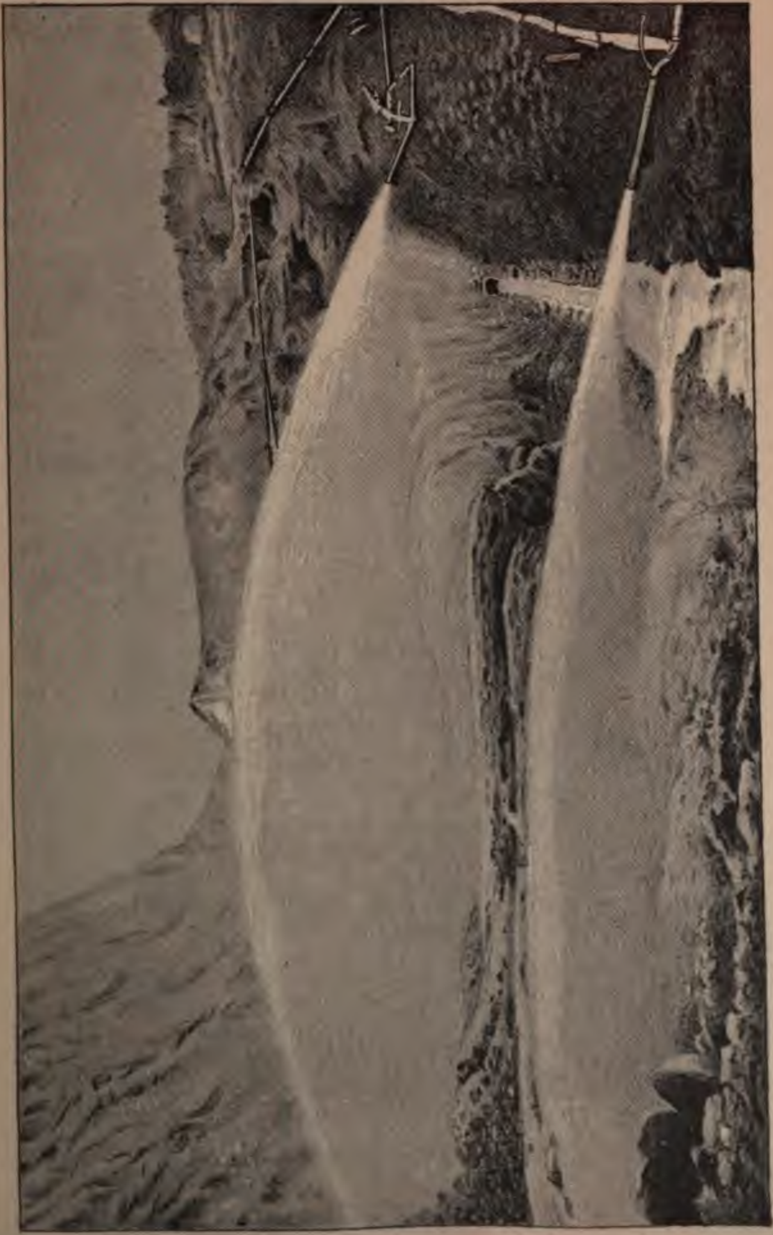

Fig. 18.

made of wood, are of more massive construction, in accordance with the great quantities of gravel to be handled and the continuous nature of the work. The sluices are commonly called "flumes," but it is better to restrict the use of this word to a conduit for carrying water only. The sluice boxes used in hydraulic mining, though, as usual, only 12 feet long, are as much as from 3 to 6 feet wide and from 2 to 3 feet deep; they are lined with heavy planks on the sides, and the pavements are made of more durable materials than is usual in shallow placer sluicing, wooden blocks, rocks, or T railroad iron being most usually employed. The wooden blocks are from 12 to 30 inches square, and from 8 to 13 inches deep. They are usually made of one of the softer varieties of pine which "broom up" under friction, and thus present a better catching surface. The blocks are cut across the grain of the wood, and are set side by side across the sluice, each row separated from the next by strips of wood to which they are nailed, while they are also kept in position by the side lining which is placed upon them. The interstices in the block pavement act as gold catchers, and are filled with small stones, or, with less advantage, allowed to fill up with gravel when washing begins. On account of the rapid wearing away of the wood, much of the gold and amalgam caught is scooped out and carried off again. Wooden block riffles last only a few weeks when in heavy work, but are easy to take up and put down again in cleaning up; they are discarded when worn so as to be only 4 to 6 inches thick.

Rock pavements are made of those boulders which are most easily obtained in the particular district. Basalt is generally used, oval stones of 15 or 18 inches long and from 9 to 12 inches thick being selected and placed on end, with a slight slant in the direction in which the current flows. They are held in place by wooden planks which divide the sluice into compartments, so that if one stone works loose the pavement as a whole is not affected. The interstices as before are filled with gravel. Rock pavements are very durable, lasting from three to six months, but require more grade to the sluice, and occasion loss of time in cleaning-up and repaving the sluices. Consequently, they are never used near the head of a sluice, where cleaning-up is a frequent operation, but are often used for the lower parts of sluices, where they sometimes alternate with block riffles, and are especially suited for tail-sluices which are only cleaned-up once a year. Rock pavements cost less than other forms of riffles.

Iron riffles, which usually consist of T-iron rails, are placed longitudinally in the sluice, closely packed side by side. They present a large amount of space available for catching the gold and amalgam, last well, present little resistance to the current (so that the grade may be low while the duty of the water remains high), and are easily taken up and put down. They are, therefore, generally used at the head of the sluices. Though their first cost is higher than that of wooden blocks, they are more economical in the end, owing to the saving of time in cleaning-up and to their longer life. Egleston * instances the results of experiments made at the Morning Star Claim, California, where three sections of sluice, each 95 feet long, were laid at a distance of 300 feet from the face of the bank which was being worked. The first section was as usual laid with wooden blocks, the second with old rails, and the third with rocks. When the clean-up was made the middle section gave 9 ozs. more of gold bullion than both the others combined. If old rails cannot be had, strips of wood bound with iron are used, but are less durable and satisfactory.

At the Blue Spur Consolidated Gold Company's plant, Gabriel's Gully, New Zealand,† where the sluice is necessarily very short, most of the stones are first separated from the gravel, and the finer material is then passed over a sluice paved with transverse angle-iron riffles, placed with the hollow side facing down stream (Fig 19, in which the arrow shows the direction of the stream); these iron riffles are placed 2 or 3 inches apart Below the section containing these riffles, there is a false bottom to the sluice, formed by an iron plate perforated with small round holes through which some of the water and the finest particles of the gravel fall on to cocoa-nut fibre matting, laid on the true bottom of the sluice. Here the fine gold is caught, the principle being similar to that used in undercurrents.

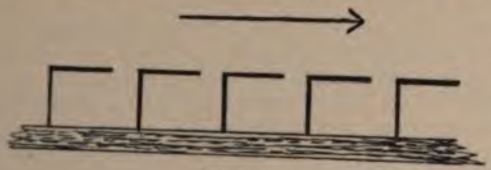

Fig. 19.
Scale = ⅛

The sluice is often divided into two by a median longitudinal partition, so that one side may be at work while the other is being cleaned-up or repaired, both sides being sometimes worked when water is very plentiful. There are usually unpaved rock-cuts above the sluice, leading to it from the places undergoing the process of piping. These rock-cuts are rarely supplied with mercury, and very little gold is usually caught there.

The length of the sluice, if capital is not lacking, depends on the cost of construction and of the maintenance, as compared

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* *Gold, Silver and Mercury in the United States*, vol. ii., p. 218.

† *Report of the Mining Commissioner of New Zealand*, 1891.

with the value of the gold saved owing to the increased length of the system. The length may be diminished by a plentiful use of drops, grizzlies and undercurrents, all of which are described above under the head of shallow placer sluicing; they are made of proportionately large size in hydraulic mining. Coarse gold is of course soon caught, but fine gold may successfully evade all the riffles of a long sluice. The Spring Valley Mine had three parallel lines of sluices, each 2½ miles in length, and it was estimated that 95 per cent. of the gold contents of the gravel was caught.* This length is unusual, the average not exceeding about 1,000 feet.

Grade of the Sluices.—The grade depends on the available fall of the ground and on the character of the material to be washed. The minimum is from 2 to 4 inches per box, such low grades being sometimes enforced by the nature of the ground, sometimes adopted from choice if the gravel is light, the gold fine, and water plentiful. With these low grades, however, disintegration is slow and incomplete; stones, unless they are small, cannot be sluiced; large ones block the sluices and must be removed by hand, and the "duty" of the water, as regards sand, is greatly decreased. The 6-inch grade is that most generally used, but as much as 12 inches per box, or even more is required when water is scarce or the gravel coarse. Steep grades effect disintegration rapidly, thus shortening the length of the sluice, and enable all but the largest rocks to be sluiced, but less gold is then caught and a more plentiful use of undercurrents is necessary. It is considered necessary to have a sufficient depth of water to cover the largest boulders to be sluiced, but the deeper the water, the more difficult it is to save gold. Where poor or top gravel is being "piped," it is worked off as rapidly as possible, and with little regard to the percentage of gold saved.

The "duty" of the miner's inch—that is, the quantity of material washed down by an inch of water in twenty-four hours—varies with the grade and other conditions. It varies from about 1 to 5 or more cubic yards of gravel per inch.

The Use of Mercury.—Mercury is added daily during the run in gradually lessening quantities, the object being to keep the mercury uncovered and clean at the top of the riffles. The feeding is regulated by the appearance of the amalgam in the sluice, the additions being made in the riffles near the head-box and in the undercurrents. The loss of mercury is usually from 5 to 10 per cent. of the amount used per run. When cemented gravels are being treated, owing to the extra amount of trituration required, the loss may be as high as 30 per cent. These losses are the more serious, for the reason that amalgam is more easily lost than pure mercury, so that a heavy loss of mercury denotes a heavy loss of gold.

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* *Ninth Report Cal. State Min.* (1889), p. 129.

Cleaning-Up.—The process does not differ from that described under the heading of shallow placer mining. It is advisable not to defer the clean-up too long as losses of amalgam are caused by the wearing of the riffles. Usually from 50 to 95 per cent. of the total yield of amalgam is caught in the first twenty or thirty boxes, which are cleaned-up frequently. The following table * shows the percentage yield of the various sections of the sluices, &c., at the North Bloomfield Mine, California, for the year 1877-8 :—

Total yield, ... $311,276.20.

Near bank, from rock cuts in mine (all in gold dust, no quicksilver being added in the rock cuts), . .	4·57	per cent.
Sluice in tunnel (1,800 feet), ...	86·26	,,
Tunnel below sluice (6,000 feet), ...	4·50	,,
Cut below tunnel (200 feet), ...	0·81	,,
Tail sluices (300 feet), ...	1·21	,,
From seven undercurrents, ...	2·65	,,
	100·00	

The first undercurrent caught five times as much as the sixth, and nearly three times as much as the seventh, which was of double size. The yield of the seventh ($947) induced the Company to add another undercurrent. This mine affords an example of the difficulty of catching fine gold. The gold loss was unknown, but was believed not to exceed 5 per cent. of the contents of the gravel.

The bullion obtained by retorting the amalgam from the sluices is finer than that from quartz mills, and is sometimes 990 fine in Australia, although Californian placer gold is often as low as 850 fine. The remainder is mainly silver, but copper, lead, iron, and some of the minerals existing in the gravel also occur. The amalgam from the head of the sluices yields finer gold than that caught lower down and in the undercurrents.

Tailings.—The tail-sluices usually terminate on the side of a cañon, in a river, or in the sea. The enormous amount of loose sand and gravel, delivered from the hydraulic mines of Placer County, California, and the neighbouring counties into the Yuba and Feather rivers prior to 1880, filled up their beds to such an extent that in rainy weather disastrous floods ensued, and much valuable agricultural land was buried beneath sterile drift deposits and rendered worthless. The farmers thereupon took action against the Mining Companies and obtained a perpetual injunction forbidding them to discharge their tailings into these rivers. The result has been to stop hydraulic mining in these districts, and the efforts to work the deep leads more extensively by drifting, or on the other hand, to impound the tailings by dams made of brushwood, or to return them to their original position, have not resulted in unqualified success. Consequently the gold winning industry has not been maintained on the extensive scale it had assumed prior to the action of the courts, the impounding of tailings behind brushwood dams having only partially solved the difficulty.

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* *Ninth Report Cal. State Min.*, p. 131.

Treatment of Cement Gravel.—In many cases the gravel from deep leads won by drift or shaft mining is cemented by iron oxides or clay into a conglomerate which is too tough to be easily disintegrated in the sluices. It is then passed through "cement mills," which closely resemble the stamp battery to be described in the next chapter, the chief differences to be noted being in the facilities for delivery. Double discharge mortars are used, and the screens are very coarse, the mesh being usually about $\frac{3}{16}$ inch, but varying up to $\frac{1}{2}$ inch in diameter. One battery of ten stamps, each weighing 950 lbs., making 94 drops of 9 inches in height per minute, will crush about 40 or 50 tons of gravel in ten hours so that it will pass though a $\frac{3}{16}$-inch mesh screen. Mercury is put into the mortar, and most of the gold is usually caught there on amalgamated copper plates, but copper plates outside the mortar are also used as in quartz-milling, and rubbers are employed to brighten the gold. If well-arranged plates are laid down, the number of sluice boxes which can be added with advantage is very small, a length of from 50 to 300 feet being used, the former limit being most common. No attempt is made to save the auriferous magnetic sands and sulphides which these conglomerates usually contain.

In the case of the Morning Star Cement Gravel Mine, Placer County, California, quoted by R. H. Richards,* the weight of the stamps was 850 lbs., the height of drop 6 to 8 inches, the number of drops per minute 95 to 100, the size of screen holes $\frac{3}{16}$ inch (round), and the capacity per stamp per twenty-four hours 12 tons.

In cases where cement mills are not required, the gravel is washed in sluices which differ little from those already described. The boxes are not more than from 18 inches to 24 inches wide and deep, and the series is seldom more than 300 or 400 feet long. Iron riffles are most in favour. Where the amount of gravel to be washed is small, or the water is scarce, the gravel is allowed to accumulate for some time and the water stored in a tank or reservoir. It is in some cases a great advantage to keep compacted gravels exposed to the air during a few months before washing them, as they "slack" and disintegrate under the influence of the weather, and subsequently are more easily treated, while for a similar reason, tailings are sometimes impounded and re-washed after some time has elapsed. The disintegration of cemented material, which has been "slacked" by exposure to the weather, is usually completed in a *cement-pan.* This is a cast-iron pan with perforated bottom, and with a gate in the side for the removal of boulders, which are mostly barren and are separated from the auriferous material by this system, instead of being crushed and mixed with it, as is the case when stamp-mills are used. In the pan, four revolving arms, furnished with plough-shares, break up the gravel, which is carried through the apertures in the bottom by a stream of water, and falls into the sluice. A pan of 5 feet in diameter and 2 feet in depth will treat from 40 to 120 tons per day, according to the nature of the gravel.

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* [illegible] 1903, p. 211; and *Can. Mng. Rev.*, vol. xv. (1896), p. 255, [illegible] on "Cost of Milling in Cement Gravel Mines."

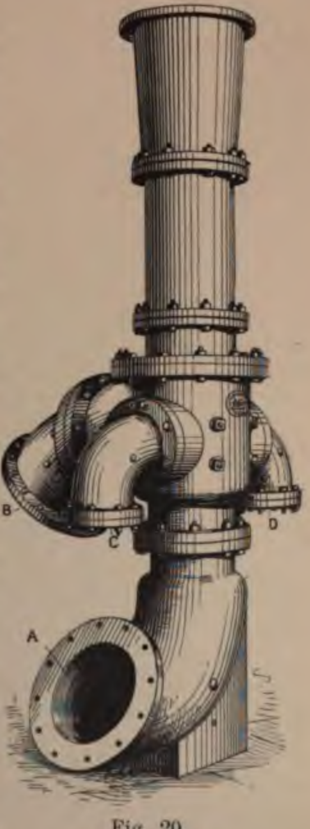

Fig. 20.

The Hydraulic Elevator.—In this machine a jet of water under high pressure forces water, gravel, and boulders up an inclined plane, and delivers them all at the head of the sluice, which may be as much as 100 feet above bed-rock. The differences in construction between the machines made in Australia, New Zealand, and the United States are only matters of detail. They consist essentially of an upraise pipe, usually of wrought iron, having a diameter of from 12 to 24 inches, which terminates below in an open conical funnel; a hydraulic nozzle, delivering water under the pressure given by a head of from 100 to 500 feet, projects into this funnel, and sand and gravel can also enter round the sides or through a special orifice. The inclination of the upraise pipe is usually from 45° to 65°. The top of the upraise pipe is turned over and terminates above a sluice, into which the gravel falls and is washed in the ordinary way. The Evans elevator made by the Risdon Iron Works, San Francisco, is shown in Fig. 20. A is the orifice through which the water for the jet is forced, B the main suction opening, and C and D two auxiliary suction openings, which can be connected with pipes of any length and serve for raising water or fine material. The upraise pipe is now generally made with heavy steel plate, and the elevator itself of cast steel, instead of cast iron, the weight being much reduced in this way. The supply pipe is greatly contracted at the throat whence the jet of water issues. The nozzle and the lower part of the upraise pump are sunk in a sump excavated in the bed-rock, and the gravel is washed down by any means (usually by a jet from an ordinary hydraulic nozzle) into this sump. The entrance to the upraise pipe is protected by a coarse grating which prevents large stones, pieces of wood, &c., from entering it. The force of water is enough to complete the disintegration of the gravel during its passage through the upraise pipe, so that a short sluice is enough to effect the washing proper. If the excavation is carefully arranged, it may be kept funnel-shaped, so that the elevator, once placed in a sump, may be worked there permanently without being moved. When the pit is large enough, the washing may be done inside it, only the tailings being raised to the surface by the hydraulic elevator. The head of water required varies according to the vertical height through which the gravel must be raised; a head of about 60 feet is required for every 10 feet of vertical upraise. Since it is just as expensive to raise water as gravel, arrangements must be made to deliver as much gravel into the sump as can possibly be raised by the jet, otherwise the expense per cubic yard will be greater, and there will be too much water with the gravel for satisfactory treatment in the sluices.

Wherever the necessary head of water is available, the hydraulic elevator is now recognised as a good method of working flat placers, river-bars, &c., or any deposits which are either below the water level of the district, or which have not sufficient fall for the disposal of the tailings by gravity. It is in wide use in California and New Zealand. The following instances of work in both countries may be given:—At the Blue Spar Consolidated Gold Mining Company, Gabriel's Gully, New Zealand,* tailings which have accumulated close to the sea on the foreshore, are sluiced in this manner. The vertical upraise is 60 feet, the angle of inclination of the upraise pipe being 63·5°; about 480 tons of gravel are raised per shift, the head of water used being 400 feet, while the amount of water used in each elevator is seventeen government heads. The sluice is short, and has an inclination of only 3½ inches in 12 feet; the upper parts are fitted with transverse, patent ┏-shaped, angle iron riffles, in which the angle faces up stream (see Fig. 19). The lower parts of the sluice have a false bottom of wrought-iron plates, perforated with round holes; beneath these plates is the [illegible] bottom of the sluice, covered with cocoanut matting in which the gold is caught. The tailings are discharged into the sea.

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* New Zealand Mining *Commissioners' Report*, 1891, p. 65.

At Quartz Valley, Siskiyou County, California,* on hard ground, where the elevator was first used, it took forty-three days to work out a piece of ground 300 feet by 250 feet, which was of an average depth of 18 feet. The bank was washed down with 600 miner's inches of water, and went to the elevator through a 30-inch bed-rock flume, which had a grade of 5 inches in 12 feet. The water and gravel were raised through a 20-inch elevator pipe without any contraction at the throat. It was set at an angle of 40°, and the pipe was 42 feet long, the vertical upraise being thus 28 feet. The force used was 1,000 inches of water with a head of 230 feet, delivered through a 6-inch nozzle, and the gravel was emptied into a sluice 6 feet by 3 feet, with a grade of 1½ inches in 12 feet. When 3,500 inches of water were running in this sluice, they could not carry off all the gravel raised by the elevator. The work was done without any delay from stoppage of the machine, and there were no repairs, the wear of the elevator being very little.

Cost of Placer Working.—In work on shallow placer deposits by individuals the results differ greatly, both according to the strength and skill of the worker and to the contents of the gravel. Under the best conditions of climate a strong, well-nourished, American digger may be able to raise by the shovel from 10 to 12 cubic yards of gravel per day, and throw it into a receptacle 3 feet above the ground. Native labour cannot be expected to effect so much, and in French Guiana it is reckoned that only about half a cubic yard of earth per man per day can be shovelled into the sluice. If the workman must wash the gravel, as well as raise it, much less can be accomplished. For an active man, it is a fair day's work to dig and wash from fifteen to twenty pans of dirt, the amount treated thus not exceeding about 10 cubic feet. On the other hand, with the cradle, the output may be from 1½ to 2 cubic yards per man in a day, while with the long-tom it may rise to 3 or 4 cubic yards per man. In the Siberian trough, only from 1 to 1¼ cubic yards can be treated by one worker per day, but a larger percentage of the gold is believed to be saved in this apparatus. The minimum contents in gold, which will make the gravel worth treating, depends, of course, on the cost of labour in the country in which the deposit exists, since few men will continue to work for less reward than they could obtain in other employments. In early times, in California and Australia, when the virgin shallow deposits were being worked, large sums were often realised by individual diggers, cases being on record in which 5 ozs. of gold were obtained from one pan of bed-rock scrapings lying under heavy gravel, and earnings of several hundred dollars per day were not uncommon. The results obtained on the Klondike in Canada are still more remarkable. Authenticated instances of 15 ozs. of gold per pan are recorded, and an average of \$5 to \$7 per pan was obtained on certain claims in 1897.

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* Egleston's *Silver, Gold and Mercury in the United States*, 1890, vol. ii., p. 307.

Concerted work, with the aid of the sluice, is much more effective; in California gravel containing about 1 pennyweight of gold per cubic yard is worked at a profit, the dirt being lifted into the sluice by hand-labour, and the tailings removed by sluicing with water; at Ballarat, in Australia, where the gravel is raised to the surface from underground workings through a vertical shaft several hundred feet deep, and subsequently washed, 12 grains of gold per cubic yard of material pay for the treatment, while in Siberia, as stated below, the cost is even less. In French Guiana, the unhealthiness of the climate and the cost of supplies render it impossible to work gravel containing less than about 3 pennyweights of gold per cubic yard.[*]

In Siberia,[†] the distance of the workings from the nearest town, and the traditions of the industry, require workmen to be hired by the year, or, in cases where no work is attempted in winter, for the season. The total cost of treatment of the gravels varies greatly with its geographical position. On the banks of the Lena, where the season only lasts for five months, it is estimated that the gravel must contain 2 *zollatniks* of gold per 100 *poods*, or 4⅓ dwts. per cubic yard; but in the neighbourhood of Ekaterineberg, the deposits in the bed of the Pechma, a tributary of the Obi, are worked when they only contain from 8 to 9 grains per cubic yard.

The cost of working the perennially frozen placers is much more, but Levat gives an instance in which a bed of gravel, 9 feet thick, yielding about 1¼ dwts. per yard, and covered by 100 feet of barren material, was worked at Malamalski in the Trans-Baikal.[‡]

The cost of hydraulic mining depends largely on the magnitude of the operations. With large quantities of water available at a cheap rate, and big banks of soft gravel, the cost has been reduced to from 2 to 3 cents per cubic yard in California, while the average cost there in 1884 was about 10 cents, and only in exceptional cases amounted to as much as 50 cents per cubic yard.

The relative costs of working the various classes of gold deposits in California by methods applicable to the respective classes, are given by John Hays Hammond,[§] as follows:—

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[*] Frémy, *Ency. Chim.*, vol. v., L'or, p. 48.

[†] Frémy, *Ency. Chim.*, vol. v., L'or, p. 47.

[‡] See *L'Or en Siberie Orientale*, vol. i., p. 146.

[§] *Ninth Report Cal. State Min.*, 1889, p. 105.

	Cost per Ton of Material treated.
1. Auriferous vein deposits (mining and milling), .	$3 to $10.
2. Drift mining, ...	75 cents to $4.
3. Miner's pan, ...	$5 to $8.
4. Rocker or cradle, ...	$2 to $3.
5. Sluicing, ...	75 cents to $1.
6. Hydraulic method, ...	1½ to 8 cents.

The cost of dredging in California may be put at 3½ to 10 cents per cubic yard of, say, 1½ tons.

CHAPTER VI.

QUARTZ CRUSHING IN THE STAMP BATTERY.

Primitive Methods of Crushing and Amalgamation.—In all countries, when the richest alluvial deposits have been worked out or have all been taken possession of, efforts have been made to extract the gold from the various hard, auriferous materials met with in veins. The earliest machines used for the purpose seem to have been stone mortars, in which the gold quartz was crushed by stone-hammers, or by large rocks raised by levers and let fall, whilst the fine material was subsequently washed out with water. The heavy particles of gold then settled to the bottom of the mortar, the lighter worthless material being washed away. Hollowed-out stone mortars, in which this operation was conducted, have been found in Wales, Egypt, Central America, the Pyrenees, and Transylvania. Diodorus Siculus, the Greek historian, has given a detailed description of the method of gold-quartz reduction employed in the mines of Upper Egypt 1,900 years ago.* The ore was first reduced to coarse powder in mortars, then finely crushed in hand-mills resembling the flour-mills of the present day, and finally washed. In order to separate the pulp from the uncrushed lumps, the Egyptians, in common with other races in ancient times, employed sieves, but in extracting the gold from auriferous sands they used raw hides, on which the flakes of gold were entangled.† These devices closely resemble those still in use in many parts of the world.‡

The date of the first use of mercury for amalgamation is unknown, but it has no doubt been used for this purpose for the last 2,000 years. The earliest mention of quicksilver itself appears to occur in the works of Theophrastus, about B.C. 300; but Diodorus, in the account just mentioned, does not refer to its use. Only a few years later, however, Vitruvius,§ about B.C. 13, described the manner in which, by the help of mercury, gold was recovered from cloth in which it had been interwoven, and in Pliny's time the separation of gold from its impurities generally by the same means was well-known.[*] It is probable that this knowledge was never afterwards entirely lost, although the references to it in the Middle Ages are very scanty. For example, Geber,[†] in the eighth century, was aware that mercury would dissolve considerable quantities of gold and silver, but not earthy materials, and Theophilus, the monk,[‡] in the eleventh century, carefully described the method of washing the sands of the Rhine on wooden tables, the final operation consisting in treating the concentrates with quicksilver for the removal of the gold. The extraction of gold by mercury is described by Biringuccio in his treatise (which was published in Italian in 1540),[§] and had apparently been a secret art for some time previously.

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* *Diodor.*, iii., 13. A full translation is given by B. H. Brough in his Cantor Lectures on Mine Surveying, *Journ. Soc. Arts*, March and April, 1892.

† Beckmann's *Hist. of Inventions*, vol. ii., p. 334.

‡ An interesting account of some primitive methods of treating gold quartz now employed by the Chinese is given by Henry Louis in the *Eng. and Mng. Journ.*, Dec. 31, 1892, p. 629, from which it appears that these methods bear points of resemblance to those of the Egyptians.

§ *Vit.*, lib. vii., cap. viii.

Mercury was introduced into Mexico as a means of extracting the precious metals by Bartolomé Medina in the year 1557,[‖] and its use doubtless soon spread to Peru and the neighbouring countries. When Barba wrote in 1639 there were three amalgamation machines in use in Peru,[¶] viz., the *mortar* (used in the Tintin process), the *trapiche* or Chilian mill, and the *maray*. The *mortar* was hollowed out in a hard stone, the cavity being about 9 inches both in diameter and in depth. The ore was triturated with water and mercury in this mortar with the aid of an iron pestle, while a stream of water, flowing through, carried away the crushed particles. The slimes were roughly concentrated, but most of the gold remained in the mortar with the mercury.

The *maray*, although equally primitive, was probably of greater capacity: it makes use of the principle employed in the bucking hammer. About the year 1825 Miers[**] saw it in Chili, where it is probably still in use. It consists of a flat or slightly concave stone, 3 feet in diameter, on which lies a spherical boulder of granite about 2 feet in diameter. This is rolled to and fro by two men seated on the ends of a long, wooden pole, which is firmly fixed to the boulder. Ore, water, and mercury are ground together in this machine, and then washed down.

The *Chilian mill* closely resembles the edge-runner mill of the present day, which is used for grinding and mixing mortar, &c. The Peruvian *trapiche* had a similar circular bed of hard stone, but only one stone runner, which was driven by mules. The

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[*] *Nat. Hist.*, lib. xxx., cap. vi., sect. 32. Quoted in full in Percy's *Metallurgy of Gold and Silver*, p. 559.

[†] Salmon's *Geber*, cap. xlvii.

[‡] *Theophili*, lib. iii., chap. xlix.

[§] *De la Pirotechnia*, Venice, 1540. Lib. ix., cap. xi., fol. 142.

[‖] Georg Agricola's *Bermannus . . . übersetzt von Friedrich August Schmid*, 1806, p. 49.

[¶] *Arte de los metales*, lib. iii., cap. xxi.

[**] *Travels in Chile and La Plata*, by John Miers, vol. ii., p. 398.

Chilian mill is still used to prepare ores for treatment in the *arrastra*, which was not mentioned by Barba, and may perhaps be regarded as an outcome of the *trapiche*.

Among other ancient methods of amalgamating ores, the *tina* and *cazo* systems may be mentioned. The *tina* system, largely used in Chili, at least as late as the year 1853,[*] is a modification of that still at work in the Tyrol (see Chap. viii.). The old and long-abandoned Norwegian method of treatment, described by Schlüter in 1738,[†] and formerly practised at Kongsberg, was also, no doubt, derived from the same source. In Norway and Chili, however, the working was not continuous. Charges of crushed ore were introduced into wooden tubs (*tinas*) furnished with cast-iron bottoms and stirred up with water and mercury by means of revolving iron agitators, comparable to the mullers of the modern amalgamating pan. At Kongsberg only 80 to 100 lbs. of ore were treated at one time with about 40 lbs. of mercury, but in Chili a *tina* contained from 4 to 6 cwts. of ore and 150 lbs. of mercury or more. After the agitators had been at work for about four hours, making sixteen revolutions per minute, the mercury was drawn off, squeezed through calf-skin or canvas bags, and the amalgam removed from the bags and retorted. The *cazo* process, introduced in Peru in 1609,[‡] was similar, but the bottom of the vessel was of copper and the pulp was boiled. These processes were used in treating ores rich in silver.

The **Arrastra** was also one of the earliest crushing machines in use in America, being introduced at the same time as the Patio process—*i.e.*, about 1557—and is still in wide use in Mexico, although chiefly in the treatment of silver ores by the Patio process. It is a circular, shallow, flat-bottomed pit, 10 to 20 feet in diameter, and paved with hard, uncut stones. Granite, basalt, and compact quartz are all used for the rock pavement, which is made 12 inches thick, and is either placed on a bed of well-puddled clay from 3 to 6 inches thick, or set in hydraulic cement, so that no chink or cranny remains into which the mercury or amalgam can find its way. In the centre a vertical shaft revolves, carrying two or four horizontal arms, to each of which is attached a heavy stone by thongs of bullock hide, or by chains. These drag stones weigh from 200 to 1,000 lbs. each, their forward ends being about 2 inches above the floor, whilst their other ends drag on it. They are moved by mules walking round outside the arrastra, or by water- or steam-power, the speed varying from four to eighteen turns per minute. Fig. 21 represents an arrastra of the simplest description; at the front the stones forming the edge have been removed, so as to expose a section of the rock pavement.

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[*] Percy's *Metallurgy of Silver and Gold*, p. 567.

[†] Schlüter's *Gründlicher Unterricht von Hütte-Werken*, p. 211.

[‡] Barba, *Arte de los Metales*. *Loc. cit. ant.*

Ore of about the size of pigeon's eggs is introduced, enough water being added to make the pulp of the consistency of cream, and mercury is sprinkled over it after most of the grinding has been done. When the ore cannot be ground any finer, more water is added to dilute the pulp, the mule is driven slowly for half an hour to collect the mercury, and the pulp is then run out and another charge shovelled in. An arrastra of 10 feet in diameter takes a charge of 500 to 600 lbs. of ore, and treats about 1 ton per day of twenty-four hours. The amount of wear sustained by the grinding stones is equal to from 6 to 10 per cent. of the ore crushed. The output is so limited that the use of the arrastra has never been general outside Mexico, although it has been used in almost every district in the United States for a short time after the commencement of quartz mining in the particular locality.

It is often stated that the results obtained, so far as the percentage of gold extracted from refractory ores is concerned, cannot be equalled by any other amalgamating appliance, and that the Mexicans, using the arrastra, formerly treated at a profit ores which hardly yield any gold to the stamp mill, or even to the amalgamating pan. In consequence of its power of saving gold and the cheapness with which it can be erected and worked, the arrastra is still valuable for prospecting. In preparing the bed for this purpose, every care must be taken that the surface is even, and all joints properly cemented, or else the mercury and amalgam will in great part find their way into the foundations.

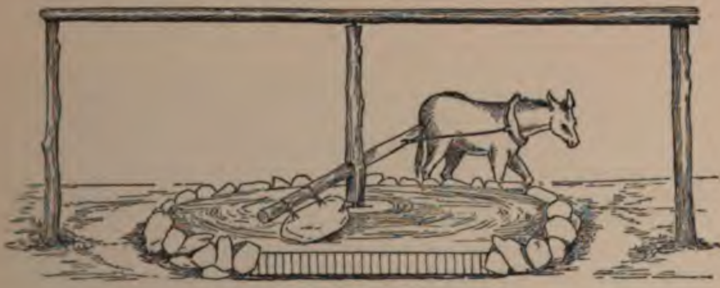

Fig. 21.

The reasons for the high extractive power of the arrastra, when treating certain ores, are no doubt to be found in the extreme fineness to which the ore is reduced, and in the prolonged contact between the ore and the mercury, which is maintained while they are being ground together. Moreover, the grinding action of the dragged stones keeps the particles of gold bright and in a suitable condition for amalgamation, without exercising force enough to flatten and harden them. The relative advantages of grinding surfaces consisting of iron and stone are less certain, and probably vary with the nature of the ore in course of treatment. An instance is given by Colonel Harris,[*] in which stone was better than iron. In this case, at Cerro de Pasco, Peru, the old method of grinding ores in circular pans, by edge-runners of stone or granite, was found to entail a rapid wearing of the edge-runners, and, in order to remedy this, the runners were shod with iron. The returns at once fell off, and on careful trial it was found that the yield in the old machines was 15 per cent. more than in the new ones. Other similar instances are on record, but it must nevertheless be conceded that, with some ores, the presence of iron is necessary for good work, chlorides of silver and mercury being reduced by it, which would otherwise be lost in the tailings. The somewhat extravagant views often expressed, upholding the great superiority of the arrastra over its more modern rivals as an amalgamating machine for gold ores, are perhaps hardly justified, since a direct comparison with stamps or pans has been possible in only a few instances.

One of the most remarkable of these instances is that afforded by the experience at the Pestarena Mines, Val Anzasca, Italy. The ore of this mine contains about 20 per cent. of pyrites, and is of somewhat low grade, rarely containing as much as 15 dwts. of gold per ton. Efforts were at first made to treat it by means of stamps and amalgamated plates, with the result that only 65 per cent. of the gold was extracted. Better results attended the introduction of the Francfort mill, a modified arrastra, driven in this instance by steam; this mill is substantially a wooden pan, with dies and shoes of stone. The mercury was added to the pulp after it had been finely ground, and the amalgamation and grinding of the pulp subsequently kept up for seven hours. From ore containing 12·3 dwts. per ton, the mill extracted 10·2 dwts., or 83 per cent., for a considerable period of time, while in the year 1890 the average extraction was 79·4 per cent. There were twenty-eight mills, each of which treated 1,200 lbs. of ore per day. The stone bed is said to last ten months, and the shoes from six to eight weeks, the total cost of treatment being very low.

At Smartsville, and in other places in California, the arrastra was applied to crushing and amalgamating hard cement-gravel obtained from a drift mine.[†] At Smartsville the cement is coarsely crushed by being passed through a Gates rock-breaker, and is then charged into four arrastras, each of which is 12 feet in diameter and 3 feet deep and capable of containing from 5 to 9 tons of gravel. The grinding is effected by four blocks of diabase, each weighing from 600 to 1,000 lbs.; the rate of revolution is 14 times per minute, the time of grinding being one hour. A tablespoonful of mercury is fed in with each charge, and the total loss is only 10 per cent. of this. The pavement costs $40 and lasts for six months. At the end of the hour, a gate is opened in the side of the arrastra and the charge run out into a sluice, 200 feet long, where the mercury and amalgam are caught by means of riffles. The capacity of one arrastra is 50 tons of hard cement per day, or 75 to 90 tons of soft surface-gravel per day. The cost is from 6 to 8 cents per ton, and more gold is extracted than when a stamp battery was used at a cost of from 20 to 40 cents per ton. One man attends to the four arrastras, and another to the Gates crusher.

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[*] *Min. Journ.*, December 24, 1892, p. 1466.

[†] *Eleventh Report Cal. State Min.*, 1892, p. 315.

Iron prospecting arrastras with stone drags are sometimes used.

There were about 100 arrastras at work in California in 1889, each treating from 1 to 3 tons of ore per day. They are used where only small quantities of high-grade ore are available for treatment.

The Stamp Battery.—The stamp battery evolved, no doubt, from the pestle and mortar was not introduced until a comparatively recent date. Beckmann * states that mortars, mills, and sieves were used exclusively in Germany throughout the whole of the 15th century, and in France stamps were unknown as late as the year 1579. Brough has suggested that the origin of stamp mills was probably due to the manufacture of gunpowder. It seems certain that in 1340 a stamp mill, used in connection with this industry, existed in Augsburg, and that Conrad Harscher, of Nuremburg, owned one in 1435. They were first applied to the gold industry at the beginning of the 16th century, a doubtful record stating that they were introduced into Saxony by Count von Maltitz in 1505, whilst in 1519 the processes of wet-stamping and sifting were established in Joachimsthal by Paul Grommestetter, who had some time previously introduced them at Schneeberg. The improvements gradually spread through Germany, and detailed descriptions and drawings of the apparatus were given by Agricola † in 1556, from which it appears that the earliest battery consisted of a single stamp, raised by means of two levers fixed to the axle of a wheel. Dry crushing was at first employed, but the great production of dust soon led to the use of water in the mortar. In some Hungarian mines, Bennett H. Brough ‡ saw some primitive stamps in use, resembling those drawn by Agricola, weighing only 100 lbs. each, and having their heads made in some cases of hard blocks of quartzite. At that time, in cases where the conditions of water supply were favourable, these stamps were able to treat with profit an ore containing as little as 2½ ozs. of gold to 50 tons of ore, and at Zell, in the Tyrol, they were able to treat a slaty material containing 1 oz. of gold to 50 tons of ore. Such economical work is seldom possible with the modern Californian stamp under the most favourable circumstances.

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* *Hist. of Inventions*, vol. ii., p. 334.

† *De re Metallica*, vol. viii., pp. 233 and 247.

‡ *Proc. Inst. Civil Eng.*, Session 1891-1892, part ii.

Agricola's exact description of the treatment of auriferous quartz in Germany in 1556 shows that the methods in use at that time were strikingly similar to those still employed in Transylvania and the Tyrol, which were among the districts of which he wrote. Doubtless in these districts, the methods have been handed down from generation to generation with little change, while in other countries, where they were introduced hundreds of years later, the changes have been rapid and striking. In those points in which the older differed from the modern Tyrolean practice, it resembled the practice of the ancient Egyptians, so that the origin of the methods may, perhaps, be traced back to them. The wooden stamps, shod with hard stone or iron, were arranged in sets of three, and raised by cams to fall by gravity when released. The rock was shovelled dry into the mortar, and coarsely crushed by the blows of the stamps.

Next, it was ground as fine as flour in a stone mill supplied with water, and carried by the stream of water into the uppermost of three wooden tubs, whence it overflowed in succession into the other two. Revolving mechanical stirrers, furnished with six paddles, kept in agitation the contents of the tubs and "separated even very minute flakes of gold from the crushed ore. These flakes, settling to the bottom, are drawn to itself and cleansed by the quicksilver (lying in the tubs), but the water carries off the dross."* Agricola here expounds the theory of amalgamation still adhered to in Austria, where mercury is regarded merely as a useful means of collecting particles of gold, which have already been separated from the crushed ore by their great density. The Tyrolean bowls, still in use at Vöröspatak in Hungary and in a few retired valleys in the Eastern Alps, do not differ essentially from the tubs drawn and described by Agricola; and, although wet crushing by the stamps has been introduced, the mortar is even now seldom furnished with screens in these mills.

Elsewhere the changes in stamp battery practice, introduced since Agricola wrote his treatise, have been many and great. One of the first was the insertion of screens in the side of the mortar, so that two operations of crushing and sifting were united. In 1767 M. Jars saw these in use in the Hartz,† though even then only a single screen of brass wire 12 inches square delivered the product of three stamps, and in several other districts in Germany screens had not been adopted. The screen was completely protected from the splash of the stamps, so that the sieving of the ore was very slowly effected by a current of water flowing through.

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* Agricola, *loc. cit.*

† *Voyages Métallurgiques*, vol. ii., p. 309. Paris, 1780.

The most important improvement, however, has undoubtedly consisted in combining the operations of crushing and amalgamation by charging mercury with the ore into the battery, and placing amalgamated copper plates in the mortar, so as to catch gold. No mention of these practices appear to have been placed on record before stamp batteries began to work in California in 1850, although they had possibly been adopted in Georgia before that time.

The use of the copper plate was probably suggested by the experience in Mexico and South America of the working of the Cazo process, in which it was well known that amalgam tended to adhere to the copper sides of the vessel unless the proportion of mercury to gold and silver present was less than 4 to 1. Thus Baron Born wrote in 1786,*—"In new kettles . . . the inside becomes wholly and so perfectly silvered that it never can be cleaned. . . . The silvery coat is daily increased by slow and gradual apposition, and the crusts of amalgama, accumulating on the bottom and sides of the vessels, become gradually so thick that on emptying them they often fall off by their own weight as silver plates, which, when dry, show the laminated texture of their daily augmentation."

The knowledge of this behaviour of amalgam in the Cazo process must have been common to many who were engaged in exploiting the quartz veins of the West soon after their discovery, and the speedy application of this knowledge is exactly what might be expected from those quick-witted pioneers. Nevertheless, the exact date and locality of the introduction of the copper plate remains a matter for conjecture.

The copper plates fixed in the battery in the early fifties were about 4 inches wide and as long as the mortar, and were placed one on the "feed" side and one on the discharge side just underneath the screens. It was soon found that the plates worked better from the start if they were coated with mercury before they were placed in position, and this has now been invariably done for the last half century. Crushed ore, stones, water, and amalgam are flung violently against the plates, and the amalgam is retained in great part.

The scouring action of the pulp on the plates is, however, always great, and becomes more violent in proportion as the stamps are larger. The plate on the feed side, long ago con.

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* Baron Inigo Born's *New Process of Amalgamation*, p. 122, translated by Raspe. London, 1791.

demned by many, is accordingly being discarded more and more, and that below the screens is curved away in such a manner that it cannot be struck directly by the splash from the stamp (see Fig. 29, p. 107). A more recent step in this direction was to line the mortar with cast-steel plates, furnished with slots for the purpose of catching amalgam (see Fig. 29).

The tendency is now towards the discontinuance of inside amalgamation, and the abolition of gold-catching plates in the mortar. The introduction of coarser crushing in the battery, followed by re-grinding, has, of course, assisted this movement.

After leaving the mortar, the pulp was treated fifty years ago, in districts other than Transylvania, mainly by passing it over inclined tables covered with blankets, much in the same way as Jason may have seen the golden sands worked in Asia Minor. The sands accumulating on the blankets were washed off at intervals and ground in mills with mercury. In addition, the ore was frequently passed over or through baths of mercury, in imitation of the old Tyrolean practice, still adhered to, with modifications, in many Australian mills.

Amalgamated copper plates over which the pulp flowed were tried, after those placed in the mortars had been proved to be beneficial, but in Western America were at first almost everywhere rejected,* probably owing to the great depth of the stream of ore and water made to flow over them. When long afterwards, about the year 1870, this mistake began to be rectified, the value of the plates was soon recognised in California.

The pulp is now led over the surface of these inclined plates in a very thin stream, not more than a quarter of an inch deep. The pulp does not run down in a regular stream, but in a series of little wavelets which tumble over and over, and are supposed to bring every part of the pulp successively in contact with the amalgamated surface. The catching powers of the plates are thus supposed to be practically independent of the tendency of the particles of gold or amalgam to sink to the bottom of the stream. This theory is not accepted by the Austrian school (see above, p. 98), and it is certain that native gold is caught more easily in proportion as it contains less silver (and is of higher density), so that when the particles of metal consist of an alloy containing a large proportion of silver, and are, therefore, of comparatively low density, the yield on the plates is generally poor. In any case, however, whatever be the working hypothesis adopted, the amalgamated plate should theoretically be better adapted for its work than the Tyrolean mill and other machines using mercury baths, owing to the slight depth of the pulp on the plates, and the short distance through which the gold particles are compelled to settle before reaching a catching surface. The plates are wiped down with rubber or brushes about once a day, and the gold separated in the usual way from the excess amalgam thus collected.

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* See *Nevada and California Processes of Silver and Gold Extraction*, p. 61, by G. Küstel. San Francisco, 1863.

The German stamp has a rectangular stem made of wood or, latterly, of iron, with an iron head, the total weight never exceeding 300 to 400 lbs. It was introduced almost unchanged into France, Cornwall, and, after the discovery of gold in 1849, into the United States, but has given place in all new districts to the Californian stamp, and need not be fully described here. In 1850, the first stamp mill seen in California was erected at Boston Ravine, Grass Valley. The stamps consisted of tree-trunks shod with iron, and the framework was constructed of logs.

The ordinary method of reduction and amalgamation of gold quartz in a stamp battery now consists of the following operations :—

1. The ore is broken down to a moderate size, usually to about 2-inch cubes, by passing through the jaws of a rock-breaker, or by hand hammers.

2. The ore is then fed into the mortar-box of a stamp mill, where it is pulverised to the required degree of fineness. In wet crushing, a stream of water is introduced also, and the blows of the stamps splash the water and pulp against screens set in the side of the mortar, the finely-divided ore being ejected in this way. In some cases the mortar-box is partly lined with amalgamated copper plates, by which some of the gold is caught and retained, mercury being in this case usually fed into the mortar-box with the ore and water.

3. On issuing from the battery, the pulp is allowed to run over a series of inclined, amalgamated, copper plates, by which a further percentage of the gold is amalgamated and retained.

4. The tailings are sometimes further treated by running over rough hides or blankets, by which some particles of gold and pyrites are retained, or the pyrites is separated from the valueless sands by concentration on some form of vanner or jig. These concentrates are subjected to further treatment, usually either by cyaniding, smelting, or chlorination. In other cases the tailings are separated, by settling in water and decantation, into "sands" and "slimes," which are cyanided without previous concentration.

5. At intervals the gold amalgam is wiped off the copper plates, the excess of mercury separated by squeezing through filter bags of chamois leather, buckskin, or canvas, and the solid amalgam thus obtained is retorted so as to distil off the mercury, and the gold is then melted.

The following is a general description of the machinery employed in stamp-battery practice :—

Rock-breakers.—There are two classes of these machines in general use, viz. :—(*a*) Jaw crushers with reciprocating motion, and (*b*) gyratory crushers. The Blake and the Dodge crushers are representative of the former class, and the Gates and Comet crushers of the latter.

The *Blake Crusher* is shown in section in Fig. 22. The rock is crushed between the stationary jaw, B C^1, and the swinging jaw, D, which is pivoted at E, and moved by the eccentric, F, through the toggles, J K. The jaw plates, C^1,C^3, usually consist of chilled cast iron with longitudinal corrugations. The machine works at about 250 or 300 revolutions per minute. At each revolution the moving jaw is advanced about ½ inch towards the other, and the lumps of rock which have dropped down between the jaws are broken; as the moving jaw recedes, the fragments slip lower down and are further crushed at the next advance, and this process is repeated until the ore is small enough to pass out at the opening at the bottom. The distance between the jaws at the bottom limits the size of the fragments, and this distance may be regulated at will by moving the wedge, L, or by changing the length of the toggles, J K. The capacity of the machine is great, being about 300 tons of ordinary rock per day of twenty-four hours in the case of the machine whose dimensions at the mouth are 20 inches by 10 inches, when the lower edge of the jaws are set to approach within 1½ inches of each other. The power required for this is stated to be 14 H.P.

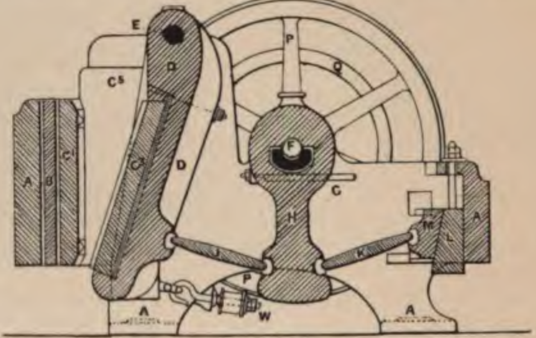

Fig. 22.

In the *Dodge Crusher*, shown in Fig. 23, the moving jaw is pivoted below instead of above. The effect of this arrangement is to make the product more uniform in size, and as there is little or no motion of the movable jaw at the delivery aperture, this may be made as narrow as desired, so that a finer product can be obtained, although it is at the expense of capacity. The Dodge crusher is more particularly recommended for fine crushing in concentration works, or where the product is to be subsequently passed through rolls.

Authorities differ as to the relative advantages of the two positions of the pivot. R. H. Richards decides* in favour of placing the pivot above, assigning as a reason that the work of crushing is in that case more evenly distributed, whereas if the pivot is below, most of the power is used near the throat or discharge aperture.

The *Gates Crusher* is shown in Fig. 24, and consists of a vertical spindle of forged steel, G, revolved by the bevel wheel, L, and a bevel pinion. The spindle is set eccentrically in the bevel wheel, and has a gyratory motion when rock is being crushed. At the top of the shaft is a chilled-iron breaking head, F, and the shell surrounding this is lined with twelve chilled-iron concave pieces. E. These form the crushing faces.

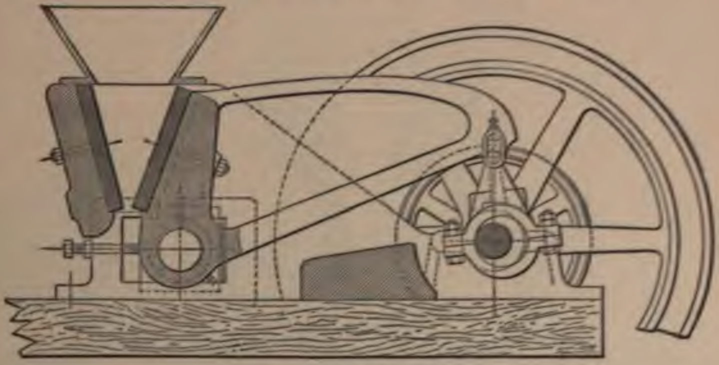

Fig. 23.

The *Comet Crusher* is a similar machine which has come into use more recently. In the later form of this machine, the "Adjustable Comet," the size of the product can be varied by turning a hand-wheel.

Gyratory crushers are of far greater capacity than machines with reciprocating motion, and have accordingly come into use where a large output is required. The largest size Comet crusher is said to have a capacity of from 100 to 200 tons of rock per hour.

Position and Use of Rock-Breakers.—The aperture of a rock-breaker is placed on a level with the floor, so that the ore can be dumped down by the side and shovelled into the jaws. It is now becoming customary to place the rock-breaker in a separate building distinct from the battery house.

A grizzly or screen of steel or iron bars, set 1 or 2 inches apart, is usually employed to separate the fine material, which is passed straight to the stamps.

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* *Ore Dressing*, vol. i., p. 35.

The efficiency and economy in crushing, attained by rock-breakers, are so fully recognised that efforts have been made to use them to reduce gold-quartz to a very small size before feeding it into the stamp batteries, with a view to increase the output. For fine crushing, multiple-jaw crushers, on the same principle as the Blake, have been constructed, but have not passed into general use; the use of a pair of rolls between the rock-breaker and the stamp battery has also been advocated.

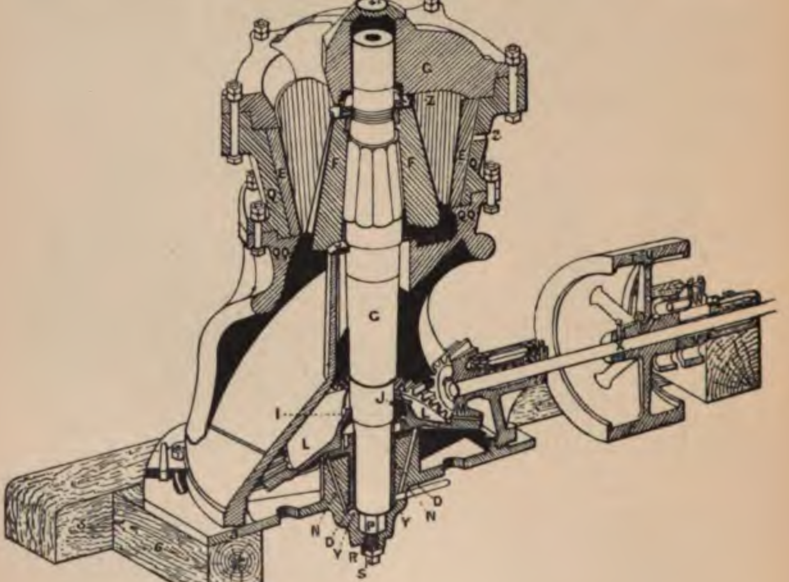

Fig. 24.

The usual size, however, to which ore is reduced before it is fed into the stamp mill is $1\frac{1}{2}$ or 2 inches maximum. Any finer preliminary crushing is unusual, and probably uneconomical. At the present day no large mills are erected without rock-breakers, which have also been successfully added to many old mills.

The Stamp Battery.—Californian "gravitation" stamps are in general use at the present day for crushing gold ores. A stamp is a heavy iron, or iron and steel, pestle, raised by a cam keyed on to a horizontal revolving shaft, and let fall by its own weight. Stamps are ranged in line in groups of five stamps each, which have a mortar-box in common. Fig. 25 represents the side view, and Fig. 26 the front view, of a ten-stamp battery, with the amalgamating tables removed to show the foundation timbers or mortar-blocks, A.*

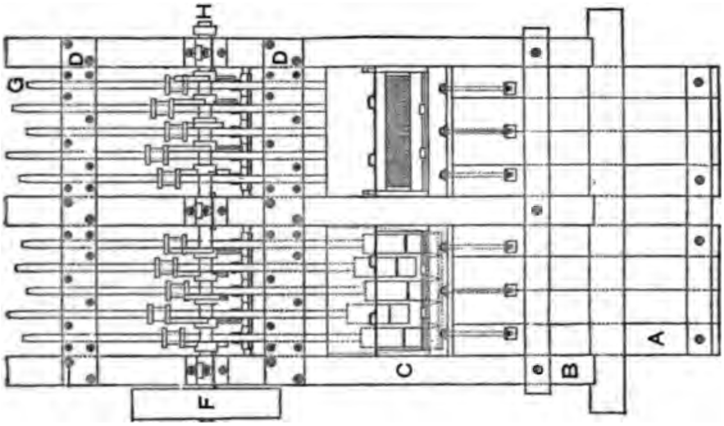

Fig. 26.

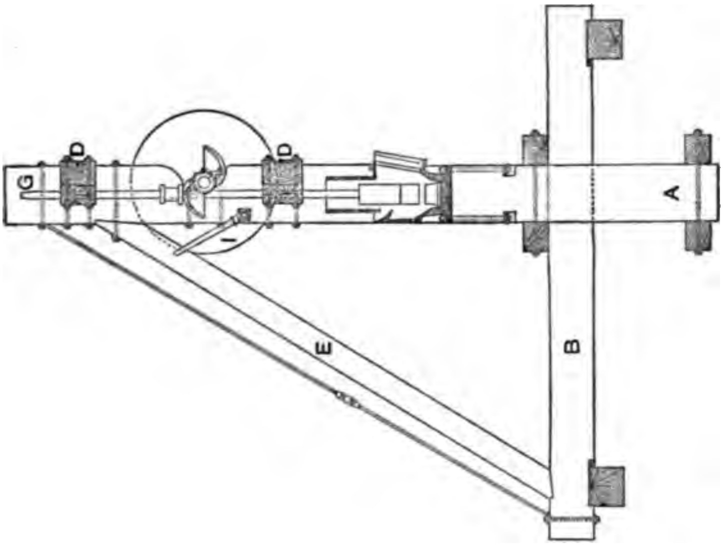

Fig. 25.

The foundations are of the highest importance, as, if they are

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[*] For full details of construction of stamp-mills, see Richard's *Ore Dressing*, vol. i., pp. 144-230.

badly made through carelessness or false economy, the efficiency of the battery is greatly decreased, and it soon shakes itself to pieces. The blow of a stamp is partly employed in crushing the ore, and is partly expended in producing a concussion or jar acting on the framework and foundations. The amount of energy used up in the latter way depends largely on construction. In preparing the ground for the foundations, the earth is removed until bed-rock is reached if possible, and the latter is then carefully smoothed and covered with a layer of concrete. The wooden mortar-blocks of from 6 to 18 feet long are placed upright in this trench, and the space round filled up with sand, or, as in the Transvaal, solid masonry is built round the blocks. Concrete mortar blocks are now often used instead of wooden ones. Heavy iron anvils or mortar blocks are also used on concrete foundations. The *framework* is made of wood, iron, or steel. It consists of the massive cross sills, B, on which rest the battery- or king-posts, C, and the braces, E. The posts are held together by the guide- or tie-timbers, D. The mud-sills are shown below the cross-sills. Frames of a number of different designs are in use.

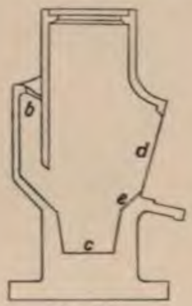


Fig. 27.

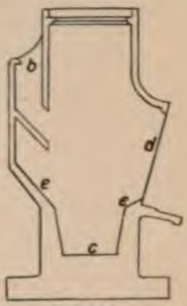


Fig. 28.

The Mortar.—The mortars are made of cast iron, but differ in shape according to the nature of the ore and the corresponding modifications made in the course of treatment. They weigh from 1½ to 4 tons, being especially thick at the bottom where there is the greatest strain. An ordinary mortar is about 4 feet 7 inches long, 50 inches high, and 12 inches wide on the inside at the level at which the dies are set. The bottom is from 3 to 11 inches in thickness, and has a heavy flange cast on it, by which it is bolted to the mortar-blocks. These are tarred over, all cracks in them having been filled with sulphur, and are then covered with three thicknesses of blanket, carefully coated with tar on both sides. The mortar is placed on these blankets and securely bolted down. This arrangement lessens the chance of the mortar working loose, the jar being diminished. A sheet of rubber, ¼ inch thick, may be used instead of the blankets. Figs. 27 and 28 represent sectional elevations of the two chief types of mortars, Fig. 27 showing a mortar intended to be supplied with an inside amalgamated copper lining plate, *e*, on the screen side only, and Fig. 28 a mortar designed to have copper plates, *e*, *e*, placed both at the front and back. In both figures, *b* is the feed-opening through which the ore is introduced into the mortar; *c* is the bed on which the die is placed; *d* is the screen-opening. The chief difference between them is in the feeding arrangement; in the latter case the back plate is put in a recess, and is protected from the falling rock fed into the battery. Back plates are now not often used. The plates catch the coarse gold inside the mortar when the pulp is flung against them.

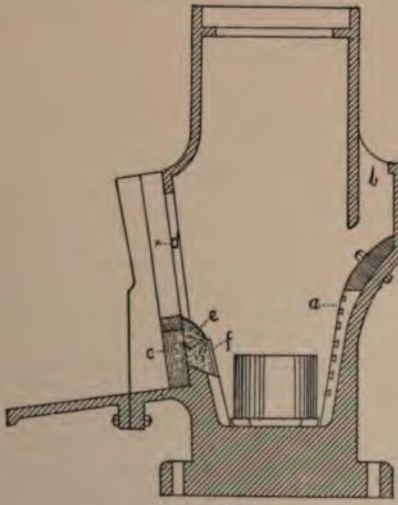


Fig. 29.

Cast-iron or steel lining plates and false bottoms take up the wear inside the mortar and are renewed when worn out. They last a few months.

A section of the mortar used at the Croesus mill, in the Transvaal, is shown in Fig. 29. *a* is a cast-steel lining plate with slots or recesses in it for collecting the amalgam; *b* is the feed opening; *c* the wooden blocks for carrying *e*, the copper plate; *d* is the screen opening; and *f* a steel plate.

The width of the mortar varies from 10 to 14 inches at the level of the bottom of the screens. As has been already mentioned, narrow mortars are best fitted for rapid discharge, but, if hard flinty ores are to be crushed, a narrow mortar causes frequent breakage of the screens, unless the discharge is deep—

i.e., unless the bottom of the screens is a considerable distance above the surface of the dies. By this latter arrangement the output is reduced, since, the nearer the screens are to the dies, the more rapid is the discharge. The depth of the discharge is the distance, measured vertically, from the top of the die to the bottom of the screen. It varies from zero to about 7 inches in modern mills. Adjustable battery-screens keep this depth constant, in spite of the wearing of the dies. The screen-frame is supported on a wooden chuck-block, which is easily removable, and to which the copper plate is bolted. When the dies wear down, the chuck-block is replaced by one of less height, to which a suitable plate has already been fixed.

The *splash-box*, not shown in the figures, and now often omitted, is bolted to the outside of the mortar just below the screens. It is rectangular, consists of wood or iron, and is of the same length as the mortar. It receives the pulp as it passes through the screens, and distributes it evenly over the amalgamating tables by a number of spouts, usually three. Instead of the splash-box, a splash-board or a canvas shield is now almost universally employed. The lip of the mortar projects some inches, as shown in Fig. 29, and to it is bolted a cast-iron *apron* about 14 inches long, which often carries an amalgamated copper plate, the *apron-plate*. The pulp is dashed through the screen against the splash-board and falls thence on the apron-plate, which is often provided with a curved lip to prevent the splashes from reaching the amalgamated tables. The old form of mortar had its upper part, or housing, of wood, but, as mercury is lost through the smallest aperture, and it was difficult to make these wooden housings quite tight, mortars are now cast in one piece, including the housings. The roof of the mortar is made of 2-inch planking, through which holes are cut to admit the stems of the stamps and the water pipes.

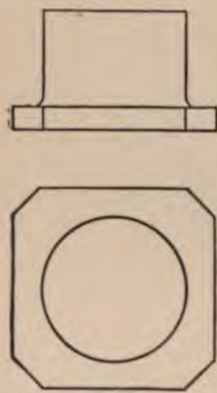


Fig. 30.

When the mortar is in place, the dies are put into it, a layer of sand being often introduced first. The dies consist of two parts, the *footplate* or *base* and the *die* proper or *body*. Fig. 30 shows, in plan and elevation, one of the many forms of dies in use; here the base is almost square, so as to fit the mortar; it is 1 or 2 inches thick, and 9½ to 10 inches square. The body is cylindrical, 5 or 6 inches high, and of the same diameter as the shoe. Shoes and dies are now generally made of forged steel, the dies being kept a little softer than the shoes. They last longer than the iron shoes and dies formerly in use. Chrome steel and manganese steel have also been used for shoes and dies. Sometimes steel shoes and iron dies are used. The wear of iron shoes and dies was formerly stated to be about 2 or 3 lbs. per ton of ore crushed in California. At the Robinson Mine, South Africa, the wear of steel shoes and dies is, according to Harland, 0·45 lb. per ton crushed for shoes and 0·30 lb. per ton for dies. According to I. Roskelley [*] a 12-inch steel shoe weighing 225 lbs. is worn out on the Rand in about ninety days, during which from 450 to 500 tons of ore are crushed. The shoe is used until it is worn down close to the shank, and weighs only 35 or 40 lbs. when discarded. Forged steel dies,[†] with cylindrical bodies 6 inches high and weighing 120 lbs., last as long as the shoes, and weigh only about 40 lbs. when thrown out, if they have worn down evenly. These rates of wear correspond for shoes to about 0·40 lb., and for dies to about 0·17 lb. per ton of ore crushed. When the body is worn down to within from ½ inch to 1 inch of the base, the die is replaced. Dies wear more slowly than shoes since they are protected by a layer of pulp, which is over an inch thick. The dies are all renewed together, as it is important that those in the same battery should be of equal height, otherwise one or more will become almost bare of ore, and a disastrous pounding result. If a die breaks, it is not replaced by a new one, but by one worn to the same extent as the others in the battery. Iron false-bottoms or chuck-blocks or sand packings are placed beneath partially-worn dies, so as to keep the depth of the discharge constant.

Dies require to be hardened and tempered very evenly, according to Roskelley,[‡] to prevent uneven wear. A die worn down at one side causes a diminution of output, and may result in a broken stem. According to E. E. Aulsebrook,[§] the three centre dies in a mortar wear down at the back faster than at the front, on account of the size of the material which is fed at the back being larger than that which is crushed at the front. For this reason he recommends dies to be turned round once a month. He also states that dies often wear unevenly because the stamps are out of centre.

The *cam-shaft*, H, Fig. 26, is of wrought iron or steel, and about 6 inches in diameter. It is now usual to have a separate cam-shaft for each five or ten stamps, which have thus a separate driving pulley. The advantage of this arrangement is that repairs can be done to one or more stamps without necessitating the stoppage of the whole mill, as used to be the case when there was only one cam-shaft. The cam-shaft is placed at a distance of from 5 to 10 inches from the stem-centre, and is 9 to 10 feet above the mortar bed. The bearings rest on supports attached to the battery posts, generally on the discharge side.

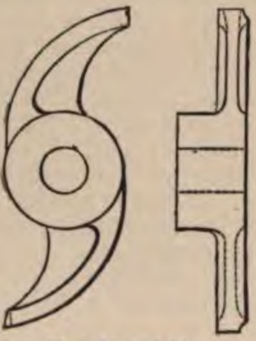
[*] *Journ. Chem. Met. and Mng. Soc. of S. A.*, Feb. 1904, p. 283.

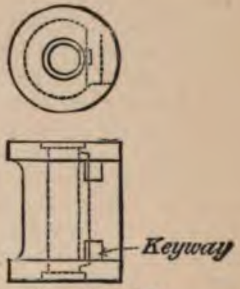
[†] *Loc. cit.*

[‡] *Loc. cit.*

[§] *Idem*, July, 1904, p. 12.

The *cams* are made of cast steel. The double cam, views of which are shown in Figs. 31 and 32, is now in almost universal use, though single and treble forms have been employed. Sometimes cams are cast in two pieces which are bolted together, so that when one is worn out, it can be replaced without first removing the other cams on the shaft, but these sectional cams work loose, and are not much used. Cams are either right- or left-handed. Both are used in the same battery to equalise the cam-thrust, the stamps being rotated in opposite directions. The shape of the cam face is the involute of a circle slightly modified at the end so as to stop the upward motion gradually. The radius of this circle is equal to the distance between the centres of the cam-shaft and the stem, which depends on the height to which the stamp is to be lifted, so that the curve of the cam varies with the drop. A cam should last several years unless broken through being a faulty casting, or through carelessness in letting the stamp fall when hung up.


Figs. 31 and 32.



Figs. 33 and 34.

The old-fashioned keyed cam has now been replaced generally by the *Blanton cam*, which will take any position on the cam-shaft. To secure it, two holes are bored in the cam-shaft, and two pins dropping into these hold fast a semi-circular tapering wedge or "bushing." The cam slips over the bushing and tightens itself in working. When it is necessary to take off a cam, a slight blow on the back edge with a hammer loosens it instantly. The "new" Blanton cam is said to be found still more convenient than the older form. The bushings are abolished, and the cam-shaft made with ten taper faces, so that its cross-section is like a ratchet-wheel with ten teeth instead of being circular. The bore of the cam has ten corresponding faces, and the cam slips on in any one of ten positions. It is then tightened sufficiently to hold it in position until put into operation, when it tightens itself further on the shaft in proportion to the work it has to do.

The cam-face works against the collar or *tappet*, shown in plan and section in Figs. 33 and 34, which is bored out to fit the stem of the stamp. The tappet is usually made of hard fine-grained cast steel, and is fitted with a wrought-iron gib, which is pressed against the stem by two or three keys behind it, thus binding the tappet firmly on the stem while, at the same time, admitting of rapid adjustment to another position. Adjustment of the tappet on the stamp-stem is required every few days to allow for the wear of the shoes and dies. The entire end surface of the tappet comes in contact with the cam-face, by which the stamp is raised and, at the same time, rotated, all its parts being round. The effect of this is, that the shoe does not strike the ore in the mortar in exactly the same place twice in succession, and the wear of its face is made more uniform. The greater part of the revolution takes place during the raising of the stamp, but the latter does not quite cease to rotate as it falls, and a slight grinding action on the ore has been noticed by many observers. The amount of rotation varies with the fall, the extent to which the cam and tappet are greased, and the state of wear of their surfaces. A little grease is always added to reduce wear, but, if too much is present, the stamp does not revolve at all, while, according to J. H. Hammond, when the tappet is in the right condition, one revolution is effected in from four to eight blows, with a 6- or 8-inch drop. Other observers find the usual rate of rotation more rapid, and in Gilpin Co., Colorado, where the average drop is from 16 to 18 inches, the stamp makes from 1¼ to 1½ revolutions at each blow, according to Rickard. Tappets last for four or five years; and, having both ends alike, they can be reversed when one end is worn out, and their worn and grooved faces can be planed down when necessary. Some millmen assert that tappets may be broken by the cam if keyed too tightly to the stem.

As the cam-thrust is not applied at the centre of the stamp, there is always a considerable side pressure, which greatly increases the friction in the guides and wears the latter out, besides causing a loss of power. Moreover, another result of this is that the stamp tends to be inclined (not vertical) when it is released, and so the blow on the die is given slightly to one side—*i.e.*, the side of the die on which the cam works. Consequently, there is a tendency for this side to wear down more quickly than the other. To obviate these disadvantages, cams have been introduced at Johannesburg with a wide hub, and the two blades set one at each end of the hub, so that they work on opposite sides of the stamp and cause it to revolve in different directions at each successive uplift. The Blanton cam, now generally used on the Rand, is fastened to the shaft by a semi-circular wedge pinned to the cam-shaft, no keys being necessary.

The pulley on the cam-shaft (F, Fig. 26) is made of wood on cast-iron flanges; if iron alone were used, the rapid succession of jars, caused by the dropping of the stamps, would soon cause the material to crystallise and break. A tightener pulley on the belt driving the cam-shaft is often used, by which the stamps can be put in motion or stopped without interfering with the driving power.

The stamp itself consists of three parts, the stem, the head or boss, and the shoe. The *stem* (G, Fig. 26) is from 12 to 16 feet long, and from 3 to 3½ inches in diameter; it is made of wrought iron or steel, or steel in the middle portion with wrought-iron ends, and has both ends tapered for a length of 6 or 8 inches to fit the heads, so that, if one end is broken off, the stem can be inverted and the other end used. The *head* and shoe are made of equal diameter—viz., about 8 or 9 inches. The head is of hard fine-grained cast steel, about 15 to 24 inches long, and has a tapered socket at each end, the upper one for the stem and the lower for the tapered shank of the shoe. When these are driven into their respective sockets, into which a few strips of wood are inserted to keep the two metal surfaces from touching each other, a few blows by the stamp bind them securely together, no other fastening being necessary. Slots are provided at the base of the two sockets, through which wedges may be driven to force out the shoe or stem when necessary. Roskelley* advocates a hole about 2½ inches in diameter drilled through the axis of the head to facilitate the removal of broken stems. The head lasts several years, being rarely ruptured. The *shoe* consists of two parts, the shank, which fits into the head, and the shoe proper or butt. It is made of hard forged steel, with the shank made softer than the butt in the tempering. The diameter of the shank is about half that of the butt. The shoe is replaced when the butt, which is from 9 to 12 inches in length when new, has been worn down to about 1 inch in length. To keep the total weight of the stamp constant, several sizes of heads are sometimes used in one mill, the heavier heads taking partly-worn shoes. "Chuck-shoes" are inserted between heads and shoes with the same object. Shoes last about three months on the Rand, according to Roskelley.

The relative weights of tappet, stem, head and shoe, which together make up the stamp, vary considerably. There is an advantage in increasing the weight of the stem, as one of small diameter tends to spring and bend from the blow of the cam, or when the stamp falls, and to wear the guides rapidly. The stem weighs from 250 to 600 lbs., the tappet from 80 to 150 lbs., the head from 175 to 370 lbs., and the shoe from 100 to 250 lbs. The total weight of the stamp is usually from 650 to 1,350 lbs., but is sometimes as low as 450 lbs., and, for prospecting purposes, the weight is only from 100 to 300 lbs.

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* *Loc. cit.*

The stamp stems are guided in boxes bolted to the wooden guide-timbers, which also serve to hold the battery posts together. There are two of these *guide-timbers* (D, Fig. 26), one within 2 or 3 feet of the top of the battery posts, and the other about 6 or 7 feet lower. The depth of each guide is about 15 inches, and the stems are fitted closely to the guides, metal boxes being used occasionally, although wood is much more general. The guide-beams are sometimes pierced with large square holes in which bushes of wood, with the grain parallel to the length of the stamp, are placed fitting the stem exactly. In this way, the guide-beams themselves are preserved from wearing out. Sectional guides, consisting of a series of iron keys enclosing wooden bushings, are also used. In this case each stem has a guide to itself and the bushings can be renewed by hanging-up the one stamp without stopping the other stamps in the battery.

Each stamp is provided with a *finger* or *jack* (I, Fig. 25) made of wrought iron, or wood protected by iron. The jack is for the purpose of raising the stamp and hanging it up out of reach of the cam. When this is to be done, a strip of wood, an inch or more thick, is laid with one hand on the cam as it rises, and the stamp is thus raised an inch higher than usual, so that the jack can be slipped in under the tappet with the other hand. The stamp is thus suspended above the cam and can be repaired without stopping the others, while it can only be released in a manner similar to that in which it was hung up. Above the stamps there is a double rail, on which is a tackle block carriage; by this the stamps, &c., can be lifted up for repairs. When the stamps are to be set up, the head is put on the die and the stem dropped into it, canvas or other packing being usually put into the head socket and the stem dropped into that. The stamp is then raised and dropped into the shoe, the shank of which is surrounded by strips of wood for packing. As already stated, the parts are soon wedged firmly together by raising and letting fall the stamp a few times.

The height of the "drop" of the stamps varies from 4 to 18 inches, and the number of drops per minute varies from 30 to over 100. These depend on one another to a great extent, an increase in the height of the drop being necessarily accompanied by a diminution in the number of drops per minute. With a drop of 8½ inches, about 95 blows can be obtained, the tappet then just having time to fall after leaving one face of the cam, before the other begins to raise it. As, within certain limits and under certain conditions, an increase of speed results in an increase of yield of pulverised ore, efforts have been made to raise the number of blows per minute. The subject will be returned to when the conditions for successful amalgamation are discussed.

D. B. Morison discusses the question of height of drop from the point of view of speed of crushing in a valuable paper, which should be consulted.* The time occupied in seconds in the fall of the stamp, if all friction, including the resistance of the air, is left out of account, is $\sqrt{\frac{2\,H}{G}}$, where H = height in feet, and $G = 32{\cdot}3$. For a fall of 8 inches this amounts to about 0·2 second, but owing to the friction between the stamp and the guides and the resistance due to the water, the actual time taken, according to Morison's experiments,† is about 0·225 second (see Fig. 35). The velocity at the time of impact and the force of the blow are correspondingly diminished, the latter being 17 per cent. less than it would be if the work were done without friction *in vacuo*, a mean result of twenty-four experiments. The time required in raising the stamp is somewhat greater, owing to the imperfect method necessarily employed. The shape of the cam causes the stamp to start upwards at a certain velocity immediately the cam meets the tappet, and to maintain the same velocity until near the end of the stroke. In a properly constructed cam, designed to give a drop of 8 inches, the vertical component of the velocity of the cam, is for ninety drops per minute, about 2·5 feet per second. If the cam were suddenly removed from the tappet when moving at this velocity, the stamp would continue to rise against gravity for about $1\frac{1}{8}$ inch, neglecting friction.

The inherent defect of the cam is that it strikes the tappet a tremendous blow in the effort to lift the stamp at the full velocity from the start, and it is this blow which is the cause of much of the intense noise and vibration felt in every stamp mill. In spite of the power of the blow, which tries the cam-shaft severely, the dotted curve in Fig. 35 shows that at first the movement of the cam is checked a little, probably in part by distortion of the cam, and that it rapidly recovers itself, and by elastic distortion in the opposite direction lifts the stamp faster than the normal rate, and these alterations succeed each other during the whole lift, after which the cam presumably recovers itself in the time that elapses before it strikes the next blow.

Fig. 35 shows a curve traced out by a pencil attached to a 900-lb. Sandycroft stamp, the ordinates representing vertical movement of the stamp and the abscissæ being time in tenths of a second. The figure was traced out during one of a number of experiments made by Morison, who subsequently devised the high-speed stamp described in Chapter viii.

Screens.—The screens are set in iron frames, which now usually slide in grooves cast in the mortar, and are keyed to it, but were formerly fitted into recesses and bolted. They are made either of steel or brass wire-cloth, or of Russia sheet-iron or steel, or tin-plate, in which holes are punched. The holes

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* "Gravitation Stamp Mills for Quartz Crushing," by D. B. Morison, *Proc. N.E.C.I. of E. and S.*, 1896-7, Session xiii.

† *Loc. cit.*

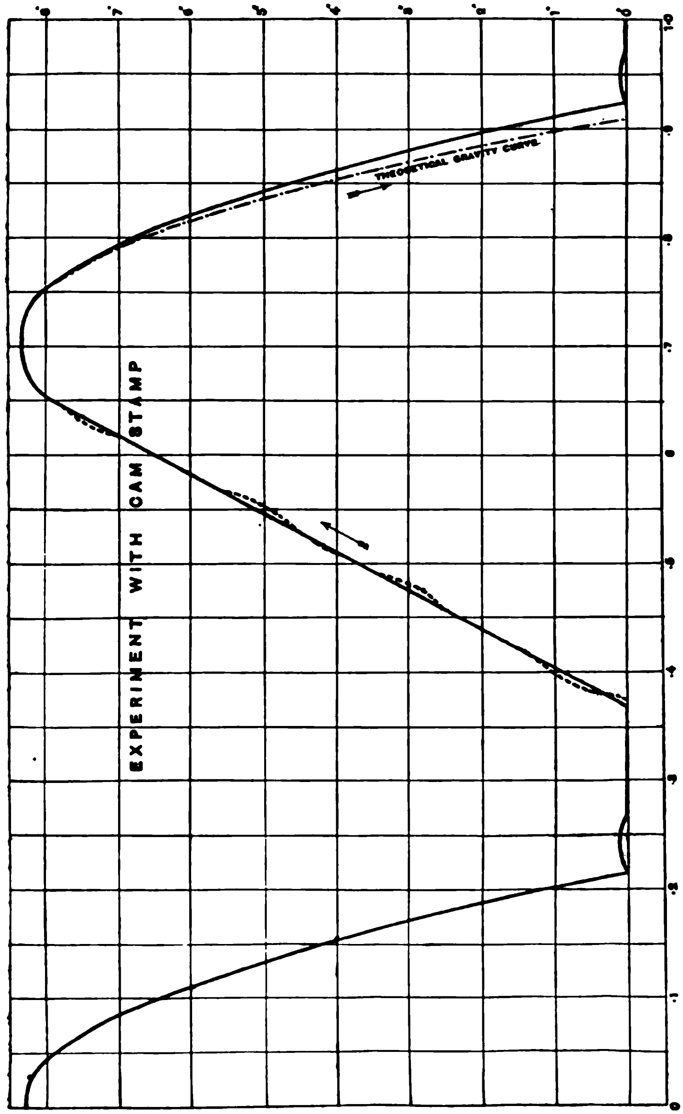

Fig. 35.—Time in Seconds.

are round, or consist of long slots (from $\frac{1}{4}$ to $\frac{1}{2}$ inch long) ranged parallel or inclined to each other. The shortest diameter of these holes ranges from about $\frac{1}{80}$ to $\frac{1}{12}$ inch or more, according to the nature of the ore and the method employed in its treatment; the usual size is from about $\frac{1}{24}$ to $\frac{1}{40}$ inch. The relative advantages of wire-cloth and sheet-iron are not yet beyond dispute, and vary with the nature of the ore. Slots appear to be better suited for discharge than meshes, but, on the other hand, there is a great loss of discharge area in the use of punched iron. Thus a wire mesh screen, containing 18 holes to the linear inch, has 324 holes to the square inch, while a round-punched sheet-iron screen has only 140 holes of the same size per square inch.

Although sheet-iron is the material usually employed for screens, it is often preferable to use copper, as pyritic ores, if kept for any length of time after being mined, soon become oxidised and acidified, and the ferrous sulphate thus formed corrodes iron rapidly, whilst the water used is often more or less acid if it comes from mines. Copper is not attacked in the same way. T. A. Rickard has shown * that the life of screens in the mills at Blackhawk, Colorado, diminishes as the creek is descended, the water becoming more and more impure. At the South Clunes United Mill, Victoria, iron punched gratings lasted less than a week, but, on introducing copper plates containing 100 holes per square inch, the life of the screen was increased to a month, and 275 tons of ore were passed through it. At Blackhawk, the iron screens last while from 80 to 430 tons are passed through them, according to the position of the mill. At Grass Valley, the average is 200 tons, at Bendigo, 134 tons, and at Otago, N.Z., only 40 tons. In California the brass-wire screens last from 10 to 14 days, corresponding to a passage of 120 to 140 tons, and the Russia-plate lasts from 15 to 40 days, the average being 30 days, corresponding to the passage of about 330 tons. The rate of wear of the screens depends greatly on their position, being more rapid with a shallow than a deep discharge, and more rapid in a narrow than a wide mortar. Pieces of iron or wood in the ore may cause the screen to break, and these should, consequently, be removed from the ore as far as possible. The battery is hung up now and then, so that a thorough inspection of all the screens may be made, and those that are broken replaced.

The screens were formerly set vertical, but they are now placed at an angle which varies somewhat but is never far from 10°, and this has been found to facilitate discharge. In wet stamping, screens are usually placed on one side of the mortar only—viz., that opposite the feeding side. In cases where the discharge is required to be as rapid as possible, the screen area is

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* *Eng. and Mng. Journ.*, Sept. 3, 1892.

increased, and double discharge (front and back) mortars have been made, but have not been used much, except for dry crushing, and screens at the ends of the mortars are used at Harriettville, Victoria (Rickard). The area of the screen is usually from 3 to 4 square feet per battery, the height from the bottom to the top of the screen being from 8 to 10 inches. On the Rand, the screen openings are usually 52 inches long and 10 inches deep. According to H. T. Pitt[*] the output is increased by about 5 per cent. by making the screens 22 inches deep instead of 10 inches. Opinions differ as to the necessary amount of screen area to be used. It has been contended that the capacity of a battery is really limited, not by its crushing, but by its discharging power. Thus, by a number of experiments conducted some years ago at the Metacom Mill, California, it was shown that when crushed pulp instead of the unbroken ore was fed into the mortar, the rate of discharge was not increased. This was taken as a convincing proof that the discharge area is usually far too small. Nevertheless, double discharge is hardly ever used for wet-crushing mills, the various objections that are made to it being summarised as follows:—

1. Inconvenience is caused in the arrangement of the copper plates, both inside and outside the battery.
2. So great a quantity of battery water must be used, that the pulp is too thin for efficient amalgamation on the plates.
3. The ore does not stay long enough in the battery to be effectively amalgamated.

Order of Fall of Stamps.—The order in which the stamps drop is of considerable importance. If they were let fall in succession from one end to the other of the mortar, the pulp would be driven before them, so that the stamp which fell last would have its die covered by too deep a cushion of ore, while that at the other end would be almost bare. The result to be obtained is to keep the ore equally distributed through the mortar, so that each stamp shall do the same amount of crushing, although it is inevitable that the middle stamps should be more efficient than the end ones in discharging the ore. The order most favoured in California is 1, 4, 2, 5, 3, and that on the Rand is 1, 3, 5, 2, 4, whilst the orders 1, 5, 2, 4, 3 and 1, 5, 3, 2, 4 are also often used. Several other orders have their advocates, and are probably little inferior to the above for the particular ores on which they are employed. Since the end stamps are of less efficiency than the others, it has been argued that a larger number of stamps in one mortar would be advantageous, and at Clausthal, in the Hartz Mountains, there are usually from nine to eleven stamps in a battery,[†] placed close together, space being greatly economised in this way. Long and wide experience has, however, proved that the best number is five.

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[*] *Journ. Chem. Met. and Mng. Soc. of S.A.*, May, 1904, p. 408.

[†] Meinecke, *Proc. Inst. C. Eng.*, Session 1891-1892, part ii.

Feeding.—Ore is fed into the battery either by hand or by automatic machines. It is often asserted that really intelligent hand-feeding is better than the automatic method, since the stamps are not all equally efficient. The feeder on small mills is often expected to break down the big pieces of ore with a sledge hammer, a rock-breaker not being used, but this method of working may be safely set down as irrational and uneconomical, and the result usually is that large and small pieces go into the mortar together. In the United States and in the Transvaal, self-feeders are almost universally employed in modern mills. The art of feeding consists in keeping the depth of pulp on the dies constant throughout the battery, as long as the work is carried on. This is much better done by automatic machinery than by hand, and it was found that by the introduction of the former in California the capacity of the stamps was increased by 15 to 20 per cent., while the wear of shoes and dies was decreased by 25 per cent., and that of the screens by

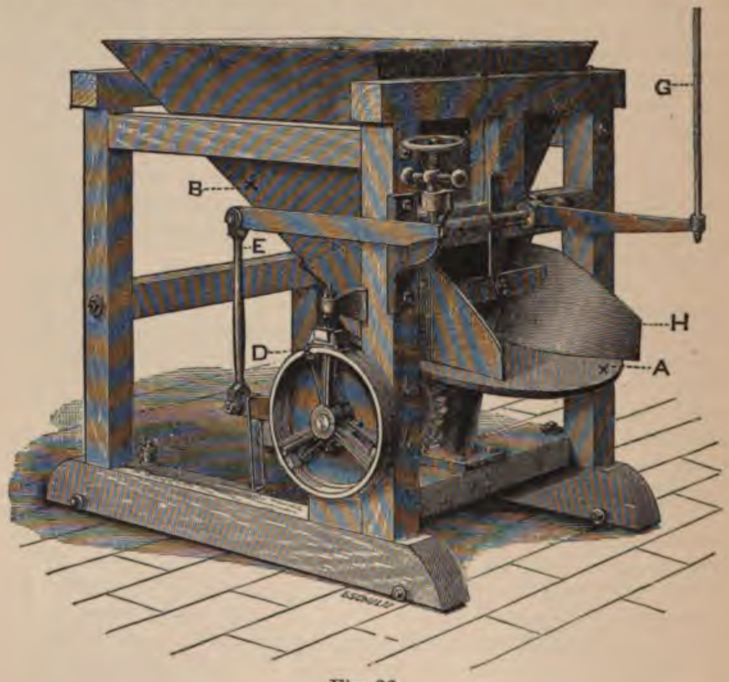

Fig. 36.

50 per cent. It is not difficult to discern the cause of the advantages, for, if the dies are insufficiently covered with ore, less crushing is done, while a greater concussion must be taken up by the stamp and by the die, mortar, &c. If the die is quite bare this concussion is so great that the stem may be bent or broken, and the shoe and die battered. On the other hand, if the ore is too deep in the mortar, there is so thick a cushion that much of the force is taken up in compression without crushing it; whilst, besides the reduction of output, the head, under these circumstances, sometimes becomes detached from the stem, which is broken or battered by the next blow. The maximum capacity is obtained with "low feeding," the depth of pulp on the dies being about 2 inches or less.

One advantage of even feeding is that a larger proportion ot gold is caught, owing to the more regular and even flow of the pulp over the plates, the danger of scouring being diminished.

There are many automatic feeders of different designs. **Hendy's Challenge feeder,** shown in Fig. 36, is a typical machine, and is more widely used than the others. It is constructed so that the tray, A, below the sheet-iron hopper, B, is revolved in a horizontal plane by means of a gear-wheel placed below it. The ore is fed into the hopper and the amount passing to the tray, A, is regulated by a sliding door. The gear-wheel is set in motion by a friction grip, D, placed on the outside of the frame, and actuated through the lever, E, by the bumper-rod, G, against which the tappet of the stamp strikes. At each partial rotation a given quantity of ore is scraped off by the stationary wings or side plates, H, resting on the tray, A. This amount of ore is regulated by the condition of the mortar. The machine is especially adapted for very wet or sticky ores. In general one feeder is sufficient for a battery of five stamps, more ore being fed to the middle stamps, where the most work is done, than to the end ones. The feeders are sometimes suspended from above instead of being supported from below.

Water Supply.—The water is usually supplied to the stamps by horizontal pipes passing just above the top of the housing of the mortar-box, with one or two feed pipes to each battery. H. T. Pitt* prefers five ½-inch holes along the back of the mortar box level with the feed chute, with the nozzles of the water pipes pointing so as to discharge the water against the dies from the back. In front of the battery there is sometimes another pipe of about half the size, to supply water to the tables to help carry off the pulp. The feed pipes are often pierced with pinholes, so that the water is supplied as a number of fine jets.

The amount of water used varies from 1¼ to 6½ gallons per stamp per minute, the average in California being about

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* *Loc. cit.*

2½ gallons, on the Rand about 5½ gallons, and in Colorado only about 1¾ gallons. In California, with fast-running rapid-discharge batteries, the amount of water per ton of rock crushed varies from 1,000 to 2,400 gallons, the mean being about 1,700 gallons, while in Colorado the average amount is as high as 2,500 gallons. On the Rand the amount of water per ton of rock is often put down as 10 tons or 2,000 gallons, but Roskelley* gives 6 to 7½ tons for the Robinson Deep in experimental runs. Besides varying with the method of crushing adopted, the amount of water varies with the nature of the gangue, clayey ores requiring more, while the large quantity required by sulphide ores is due to the deep discharge necessitated by the difficulty of catching the gold, as well as to the high density of the pulverised material, which renders it more difficult to convey in suspension over the plates. As a rule, the more rapid the output, the less water per ton of ore is required in the battery. Coarse crushing requires less water in the battery, but, on the other hand, more has to be added on the plates. The amount of battery water per ton is increased by over 20 per cent. by the employment of double-discharge mortars. The amount of water to be added on the plates varies with their grade, as well as with the density and size of the particles of crushed ore. It should be only just enough to prevent the pulp from accumulating on the plates, as any excess over this tends to check amalgamation and to scour the plates. Richards gives† the average amount of water used in 21 mills as 2·77 gallons per stamp per minute, or 6·68 tons of water per ton of ore crushed. The average duty of a miner's inch in a gold stamp-mill is given by P. M. Randall ‡ as 12 tons of quartz if the head under which the water is supplied is 4 inches, and 15·88 tons if the head is 7 inches. This gives the proportion of the volume of water to that of ore as 11·1 to 1. This may be compared with the proportion of between 7 and 10 to 1 in Siberian placer working, and as much as 30, or even 50 to 1 in hydraulic mining. It may be mentioned that a ton of 2,000 lbs. of quartz occupies about 13 cubic feet when unbroken, and about 20 cubic feet after having been broken up, so that in a lode a cubic yard contains about 2 tons, and in tailings heap only about 1⅓ tons.

Amalgamated Plates.—These plates are usually made of copper, and are as much as ⅜ inch thick for the inside of the battery and 1/16 to 3/16 inch thick for the outside. The average weight used in California is 3 lbs. per square foot. It was formerly laid down as a general rule that the heaviest plates were the best, as they last longer and are not so easily dented, but comparatively light plates are now used. The copper should

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* *Proc. Chem. and Met. Soc.*, February, 1904, p. 284.

† *Ore Dressing*, vol. i., p. 222.

‡ Article on Practical Hydraulics in *Sixth Report Cal. State Min.*, 1886.

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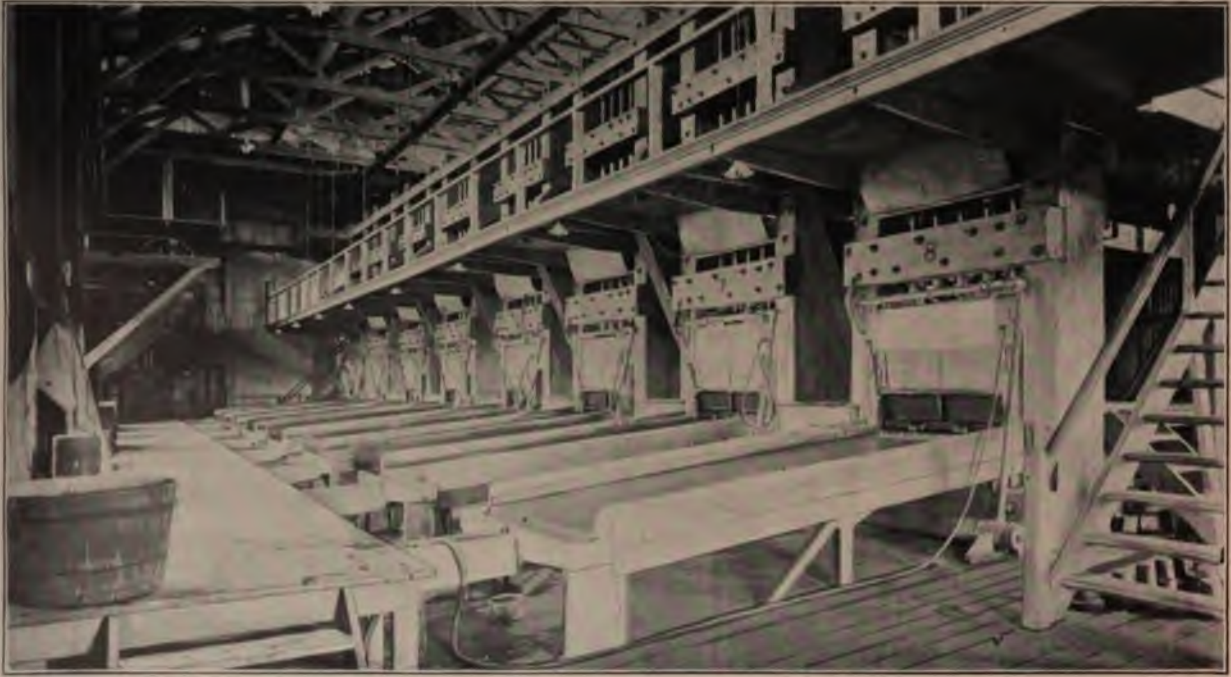
PLATE IV.



The 40-Stamp Battery of the New Heriot Company, showing the Stamps, Water Supply, and Copper Tables.

be of the best quality, and, if it is hard, it must be annealed before applying the mercury, so as to make it absorbent. This was formerly done by heating it from below as uniformly as possible until sawdust laid on the upper surface was ignited. The plate is then straightened by blows of a wooden mallet, striking a block of wood laid on it, and the surface is carefully cleaned by scouring with sand or fine emery paper until quite bright, and washed with strong soda to remove all traces of grease. Cleaning may also be effected by nitric acid diluted with 9 parts of water, or by a 2½ per cent. solution of potassium cyanide, rubbed on with a woollen rag and carefully washed off with water. As soon as the plate is clean, it is rubbed with a mixture of fine sand, sal-ammoniac and mercury by means of a brush, the sal-ammoniac preventing the recommencement of oxidation. Potassium cyanide is now more often used than sal-ammoniac, and the sand is sometimes omitted. More mercury is sprinkled on and wiped over with the brush or a piece of rubber until the surface is pasty. Care is taken not to make the plate "wet" with mercury. If any drops of mercury appear on the plate losses occur through scouring.

If the plates were used in this condition they would not catch the gold very well at first, but would continually improve until they had become coated with gold amalgam. In order to make them efficient from the start, they are usually coated with gold amalgam or silver amalgam before being laid down in the mill. Gold amalgam is most effective but is seldom used, as it is so much more expensive than silver. The amalgam is rubbed on with a piece of india-rubber, the plate being wetted with a solution of sal-ammoniac or potassium cyanide to keep it bright. The whole operation is called *setting* the plates. According to Roskelley* 16 ozs. of mercury and 6 ozs. of gold amalgam are enough for setting a plate 16 feet by 4 feet 9 inches. A more usual method of preparing the plates in California is to coat them with electro-deposited silver. This plating is done by certain establishments in California for most of the mills, but it can be done on the spot without much difficulty, the plant required being inexpensive. After being silvered, the plates have the mercury applied to them. They absorb a large amount of mercury, catch gold well, and are little trouble to keep clean. The plates need not be re-silvered, except after scraping and sweating (see under "The Clean-up," Chap. vii.), as they become coated with gold amalgam in the course of time. About 1 oz. of electro-deposited silver is required per square foot of copper plate. Silvered plates are not used inside the mortar.

The position of the plates is as follows:—The lower edges of the inside plates are level with the upper surface of the pulp, when the battery is working properly—*i.e.*, they are usually at

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* *Journ. Chem. Met. and Mng. Soc. of S.A.*, Feb. 1904, p. 286.

1½ or 2 inches above the surface of the dies. The plate on the feed side is generally about 9 to 12 inches wide, and is of the same length as the battery; it is bolted to the mortar itself, and its angle of inclination varies with the shape of the latter, so that the angle of inclination is sometimes 40°, and it is sometimes nearly vertical. The plate on the discharge side is inclined at an angle of 10° to 50° to the vertical, and is as wide as the space below the screen permits, being usually from 3 to 6 inches wide. Sometimes the plate is curved to fit the chuck-block and sometimes it is bent. It is fixed to a wooden chuck-block, which has its top bevelled off so as not to obstruct the screen opening. The block is bolted to the mortar with some thickness of blanketing between, in order to make a tight joint. Several sizes of these chuck-blocks, with their copper plates attached, are kept in one and the same mill, a wider block being substituted for a narrower one when the wear of the dies has proceeded to a certain extent.

At the Alaska Treadwell Mill, and at one or two mills on the Rand, the cast-steel linings of the mortars were furnished with several horizontal slots or recesses for the collection of amalgam, and these took the place of the back copper plates. The front or chuck-plate was, however, retained in these mortars. This arrangement is shown in Fig. 29. Inside plates are now generally discarded on the Rand.

The outside plates are fastened to a wooden table with copper nails, or wooden clamps, or by wedges driven into the raised edges of the table. The table is as wide as the battery (4 feet 9 inches) and usually from 6 to 8 feet long. On the Rand they are 12 to 16 feet long. In California a length of 2 or 3 feet of plates of the same width, the *apron plates*, are interposed between the battery and the tables proper, on to which there is a drop of 2 or 3 inches. Below the tables there is a succession of four or five *sluice plates*, each about 30 inches long, with drops of 1 to 3 inches between them. They are usually made narrower than the others, and are frequently only 12 or 18 inches wide. This practice is doubtless due to the traditions as to the treatment of auriferous gravel in sluice boxes existing when amalgamated plates were first devised and introduced. It is not to be commended, as the stream of ore and water, forced into a narrower channel, becomes deeper and flows more rapidly and tumultuously, with the result that the contact between the ore and amalgamated plates is much reduced, and very little gold is caught. The use of the drops is to assist in catching the float gold and to separate the amalgam which has become floured and mixed with the pulp. In place of the composite arrangement described above, a single length of 12 or 14 feet of plates is often used, especially in the Transvaal, but it is usual in this case for a plate about 14 inches wide to be placed at the lip of the mortar, resting on the lower bar of the screen frame. This plate corresponds to the apron plates.

To aid in catching the float gold, swinging amalgamated plates have been introduced, and are in use in the sluices below the batteries of many Californian mills. They are also used in hydraulic mining. The swinging plate consists of a curved strip of silver-plated amalgamated plate about 3 inches deep, and of the same width as the sluice in which it is hung; it is suspended on eyes through which wires pass. The plate thus hangs, half submerged, with its concave side up-stream, and is kept swinging by the current, so that all floating particles of gold must come in contact with it. It is found in practice that, immediately under each plate, across the sluice, a line of amalgam which has dropped from the plate accumulates. The plates are placed a few feet apart. They cost little and are very effective.

Mill Sampling.—It is usual to take samples of the pulp issuing from the screens, and also after it has passed over the amalgamated plates. The method adopted at some mills* is to catch the whole stream falling from the lip of the mortar for a fraction of a second. This is done by a trough 6 inches wide applied by hand. Automatic sampling would be better.

Mill Site.—This should be easily accessible by road, rail, and water, if possible; moreover, it should be near both wood and water, and there should be a good fall of the ground. The least fall that is considered sufficient in California is 33 feet from the mouth of the rock-breaker to the floor on which the concentrators are placed, when rock-breakers are used, followed by stamps, copper tables, sluice plates, and two successive concentration tables. If a second concentrator is dispensed with, however, and space otherwise economised as far as possible, 29½ feet may be enough.

Arrangement of the Mill.—The general disposition of the machinery is shown in section in Fig. 37. This represents a mill in which the ore is delivered from the ore-cars through a grizzly on to the rock-breaker floor, and thence by a shoot to the automatic feeders of the stamp battery; the pulp, after passing over the plates, is conveyed by sluices to the double row of "frue vanners" (described in Chap. ix.), which are shown standing back to back on the lowest floor. Large bins for mixed ore as received from the mine are sometimes installed of sufficient capacity to hold several days' supply for the stamps, in case of breakdowns or other delays in the mine. Their floors are made of planking, which is laid with the lengths in the direction of the slope, for, if placed transversely, the boards wear fast, and the ore packs at the edge of each one, with the result that its movement is impeded and must be assisted by shovelling. The slope should be at least 45° in order to enable the ore to move downwards by gravity, when the lowest portion is drawn from the shoot. Ore-bins with flat bottoms have greater capacity, but necessitate an additional handling of the ore. The sills of the bins should be placed horizontally on

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* *Op. cit.*, May, 1904, p. 412.

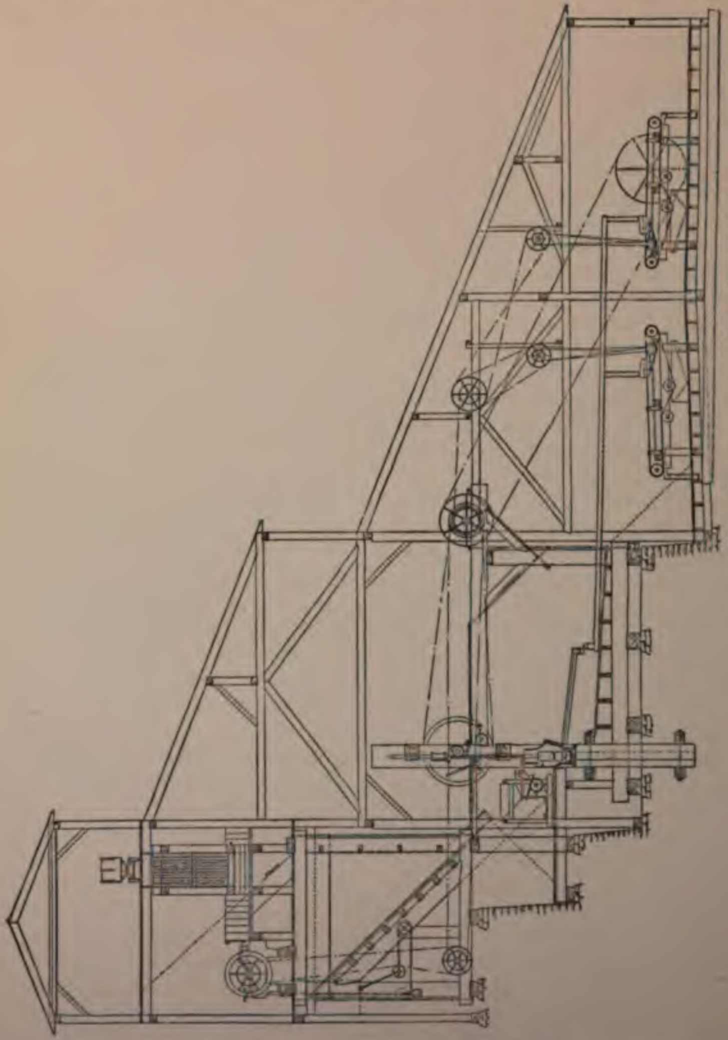

Fig. 37.

Scale, ⅜ inch = 12 feet.

terraced ground, not on the slope of the hill. A shoot from the mixed ore-bin door leads to a grizzly, through which the fine ore drops into the main battery ore-bin. The larger pieces of rock are discharged either into a coarse ore-bin or else upon the platform by the side of the rock-breaker and on a level with its mouth, into which it is shovelled by hand. The former course is preferable, as in that case the rock-breaker can be fed continuously by a gate in the coarse ore-bin, which is opened and shut by a rack and pinion. By this arrangement there is a saving of labour, but the chief advantage is that the rock-breaker is thereby kept constantly at work. At the North Star Mill, California, it was found that when, by arranging for a continuous feed from the coarse ore-bin down a shoot leading direct to the rock-breaker, the latter was in constant work, it absorbed 12 horse-power, as against 8 horse-power when in intermittent work, but its output was over 50 per cent. more. When the stamp battery is used only to crush the ore, which is subsequently treated in pans or other amalgamators, or by concentration, it is of great advantage to separate the fine product of the rock-breaker by sieving, instead of passing the whole through the stamps. This arrangement increases the output and prevents unnecessary sliming of the ore, thus greatly reducing the loss of sulphides when an attempt is made to save these by concentration. It is becoming customary in large mills to place the rock-breakers in separate buildings.

The product of the rock-breaker is mixed as thoroughly as possible with the ore, which originally passed through the grizzly, and led by means of a shoot direct to the automatic feeders, which should be mounted on wheels, so as to be readily movable. Plenty of space must be left behind the feeders for convenience in repairing and in the exercise of supervision. The amalgamating tables should also be easily accessible, space being left to pass between them, and the same remark applies to the sluices and the tables or other appliances for concentration. H. T. Pitt* has suggested that the amalgamated tables should be separated from the battery by a space of several feet to allow space for sampling and for repairs. The pulp would then be delivered at the tables by launders. All shafts, bearings, &c., should also be easily accessible, so that oiling, relining, and repairs may be readily done. The battery house should be of ample size to allow space for handling cam-shafts, &c., when they are being replaced.

The tailings are discharged into a large sluice by which they are carried into a river, or into the sea, or run into settling pits, or impounded behind dams. One of the two latter courses is adopted, either if water is scarce, so that it is necessary to use it over again, or if the discharge of tailings is forbidden by law, or if the tailings are rich enough to be subjected to further treatment, at once or at some future time.

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* *Idem*, p. 409.

The whole of the machinery is contained in a strong building of timber or corrugated iron, to protect it and the workmen from the weather, and to prevent theft of amalgam. The interior should be lighted by as many windows as possible, in order to facilitate superintendence and repairs. In cold climates, the mill buildings are sometimes warmed by passing the flues from the boiler fires through them from end to end, before leading the products of combustion to the stack, or by steam pipes.

CHAPTER VII.

AMALGAMATION IN THE STAMP BATTERY.

Treatment of the Plates.—In order to keep the plates in proper condition so that successful amalgamation may be maintained, they must be prepared carefully, and the closest watch kept over them. The setting of the plates has already been described on p. 121. The silver-plated copper table is preferred in California from the ease with which it is kept clean, but is not used in the Transvaal. It is not considered desirable to put on it as much mercury as it will hold, since, if the amalgam is too fluid, losses are sustained by scouring, but, on the other hand, if the amalgam becomes too hard and dry from absorption of gold and silver, further amalgamation is checked and fresh mercury must be added.

The condition of the inside plates is regulated by the amount of mercury supplied to the mortar. In Colorado there is an opening at the front of the battery and above the screen frame, ordinarily covered by canvas, which can be lifted up by the millman, who introduces his arm, and determines by passing his hand over the front plate, whether the right amount of mercury is being added by the feeder. The regulation of the addition of mercury is thus effected without removing the screen frame. Mercury is sometimes added direct to the outside plates, and sometimes their condition is regulated by the additions of mercury in the mortar box. The amount added varies with the conditions of crushing and the richness of the ore, but in general from 1 to 2 ounces of mercury are fed in for every ounce of gold contained in the ore. The finer the state of division of the gold and the more sulphides there are contained in the ore, the more mercury is required. It is fed into the battery at stated intervals of from half an hour to an hour. In some mills amalgamation in the battery is not attempted, no inside copper plates being provided, and under these circumstances it is not usual to feed mercury into the battery.

The practice of feeding mercury into the battery, although still frequently pursued, meets with opposition from certain experienced millmen. The objections urged against it are mainly that the mercury so introduced, and the amalgam formed through its agency, tend to become so excessively subdivided that a high percentage is lost through flouring; moreover the mercury is liable to sicken when the ores contain sulphides. These evils, no doubt, exist, and tend to increase with the percentage of sulphides present, while arsenic and antimony in particular cause heavy losses of both mercury and amalgam if battery amalgamation is attempted, but with ordinary free-milling ores such losses do not appear to be serious. When coarse free gold is present, inside amalgamation is probably advantageous, but otherwise outside amalgamation is now generally preferred.

The amalgamated plates are *dressed* as frequently as is necessary, the length of time allowed to elapse between two operations depending partly on the richness of the ore. To dress the plates, the battery is stopped, and the "black sands" are swept off and kept separately for grinding with mercury. The plates are then rubbed with a hard brush so as to soften and distribute the amalgam, and, if necessary, fresh mercury is added. The amount of mercury put on the plates should be enough to keep their surfaces in a pasty condition, but not enough to gather into liquid drops or to run off. Sometimes cyanide is added in dressing the plates, to assist in the removal of stains, but other chemicals are now seldom used. The plates are dressed once or twice a day, or as often as every two hours when rich ores are being treated, or when stains tend to appear owing to the nature of the ore. Usually in dressing the plates no amalgam is removed, but sometimes a partial clean-up is effected at the same time, the surplus amalgam being wiped off with a piece of rubber.

On the Rand* mercury is sprayed on about every eight hours to the upper 3 feet of the plates, the part that is scraped daily. The mercury oozing downwards supplies the lower part. Then a solution of 0·08 per cent. cyanide is sprinkled over the plate, and the mercury is rubbed in with a hard brush. Afterwards the plate is brushed smooth with a soft brush, and then washed down.

Whisk brooms are often used for brushing the plates; these brooms are cut down to a short length so as to be stiff enough. The plate is brushed all over and the amalgam thus thoroughly loosened from it, after which, commencing at the top where the amalgam is thickest, it is subjected to a systematic stiff brushing, each stroke being directed longitudinally down the table, and not towards the centre. The surplus amalgam is thus brushed to the lower end of the plate, whence it is removed, and a thin coating of amalgam is left over the whole surface of the plate, excluding the air and preventing the formation of "verdigris."

Discoloration of the Copper Plates.—The plates often become stained by the formation on them of oxides, carbonates, or other compounds of copper through the corrosive action of the water and pulp. Ores containing decomposing sulphides acidify the water and thus cause the corrosion of the plates, a yellow film being formed on the surface of the metal. The presence of carbonic acid in the water is equally harmful, but Aaron points out that the addition of slaked lime to the water neutralises the acid substances and diminishes the tarnishing. The yellow, brownish, or greenish discoloration, the so-called "verdigris," appears in spots and spreads quickly, especially on new plates, those which have been silver-plated being less liable to become dirty than the others; whilst when a plate has become covered with a thick layer of amalgam it is not readily discoloured. When these stains appear the plate must be at once cleaned, as the stained part catches little or no gold. The chemicals used for the purpose are generally sal-ammoniac and potassium cyanide, the operation being conducted as follows:—The battery is stopped, the plates rinsed with clean water, and a solution of sal-ammoniac applied to the stained parts with a scrubbing brush, and left covering them for a few minutes in order to dissolve the oxides. It is then washed off, a solution of potassium cyanide rubbed on to brighten the plate, and almost instantly washed off, fresh mercury being then added if necessary. Janin states (*Mineral Industry*, 1894, p. 332) that long brushing with potassium cyanide is necessary, as otherwise the spots reappear when the water is turned on.

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* I. Roskelley, *Chem. Met. and Mng. Soc. of S.A.*, Feb. 1904, p. 287.

Chemicals Used to Promote Amalgamation.—The use of chemicals to aid amalgamation was formerly much more general than at present, although battery-men were less afflicted by the rage for nostrums than pan-amalgamators. *Potassium cyanide* is used to clean the plates and to promote amalgamation, its action consisting in the removal of oil, grease, and base metallic oxides. 1 or 2 lbs. of potassium cyanide are said by J. H. Hammond to be enough to supply a 40-stamp battery for twelve months when treating free-milling ores, by which the mercury is not much affected. The difference between such a mill and one running on base ores may be judged from the fact that at the Hidden Treasure Mill, Colorado, where there are seventy-five stamps, 260 lbs. of cyanide were used in a year. Here the plates are dressed every twelve hours with a weak solution containing 2 ozs. of cyanide in 3 gallons of water, the operation being necessitated by the acidity of the water which comes from the mine, and is further contaminated by sulphates formed in the ore. Roskelley* used ½ oz. of cyanide dissolved in 4 gallons of water—that is, a solution containing 0·08 per cent. of cyanide—for dressing plates at the Robinson Deep Mill, Transvaal, and states that that amount is enough to dress ten tables. The tables were dressed every four hours. He found that it saves time to use cyanide in dressing and setting the plates, and keeps the plates in better condition by preventing the appearance of discoloured patches. D. J. Peplar† and others consider that discoloration is best prevented by prolonged rubbing with or without cyanide, and that when cyanide is used less rubbing is done, so that, although time may be saved, it is at the expense of efficiency. Potassium cyanide is sometimes added to the water used in the mill. (See below under "Crushing with Cyanide Solution.") It is said to make the amalgam on the plates hard and dry.

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* *Op. cit.*, p. 285.

† *Op. cit.*, May, 1904, p. 401.

Sodium amalgam is used to revivify sickened mercury, or to maintain it in good condition. It is prepared by heating a basin or iron flask of mercury to about 300° F., and dropping in little pieces of sodium not larger than a pea, one by one. Each addition causes a slight explosion and a bright flash of flame. The sodium may be added with less loss and less danger to the operator if the mercury is kept at a somewhat lower temperature and the sodium stirred into the mercury with an iron pestle or pressed below its surface with a spatula. When about 3 per cent. of sodium has been added to the mercury, the reaction becomes less active and the amalgam is then poured out upon a slab or shallow dish, allowed to cool and solidify, and then broken up and kept in stoppered bottles under naphtha. When it is necessary to revivify a quantity of mercury, a few small pieces of the amalgam are added to it and stirred in, or are previously dissolved in clean mercury before being added to the impure stuff. The strong affinity of sodium for oxygen, chlorine, &c., enables it to reduce the oxides and other compounds of the base metals which are coating the mercury globules. The sodium hydrate passes into solution, neutralising part of the acidity of the water at the same time. The base metals are redissolved by the mercury which is then in good condition to take up the precious metals or to be caught on amalgamated surfaces or in riffles, but the mercury is not really purified and the base metals in it are soon oxidised again. Sodium amalgam is not much used except in amalgamating-pans or in mercury wells or riffles, or in cleaning retorted mercury—*i.e.*, wherever large bodies of mercury can be directly acted on by it. It is of comparatively small value when added to the mortar of a stamp battery, although this use of it is not unknown. The use of electric currents, galvanic couples, &c., already referred to, has an effect similar to that of sodium amalgam.

The merest trace of any kind of grease or oil is very prejudicial to successful amalgamation, forming a film over the plates and over the little globules of mercury, and thus preventing contact between them and the particles of gold. Grease may consist of the tallow dropped from the miner's candles, or the oil from the loose steam which is sometimes used to warm the battery feed-water, or from the bearings of the machinery. Acidity of the water and the effects of grease are often corrected by the addition of solid *lime* to the ore. This has now become a common practice, but its efficacy is denied by Bevington,* who states that it does not remove discoloration from plates, and, moreover, tends to make the amalgam strip off.

Slope of Plates.—The grade or inclination of the plates varies with the nature of the ore to be treated, heavy pyritic ores requiring a higher grade than light quartz, while the coarser the crushing the steeper must be the grade. In California the copper tables have an inclination of from 1½ to 2 inches per foot, the apron plates from ½ to 1¾ inches per foot, with an average of 1½ inches, and the sluice plates from 1¼ to 1½ inches. The narrower plates have a lower slope in order to avoid increased scouring action. With heavily sulphuretted ores a grade of from 2 to 2½ inches per foot is used. On the Rand a slope of 1¼ to 1½ inches per foot is generally used. The steepness of the grade is of great importance, as on it, and on the amount of water supplied, the attainment of the necessary contact between the ore and the plate depends. When the pulp is flowing properly, it travels down in a series of little waves and ripples, and, in consequence of the friction between the plate and the film of water in contact with it, the upper portions of these little waves travel faster than the lower parts, so that the motion becomes one of tumbling over and over. As a result of this, if the plate is long enough, every particle of pulp comes in contact with the amalgamated surface, and the perfect extraction of amalgamable gold, mercury and amalgam is obtained.

Muntz Metal Plates.†—The use of Muntz metal (which consists of copper 60 per cent., zinc 40 per cent.) for amalgamated plates is of great interest. It differs from copper in catching gold well as soon as the plate is amalgamated, not requiring to be covered with gold- or silver-amalgam before it begins to do good work. Moreover, the amalgamated surface is very superficial, since the mercury does not sink in so far as it does into a plate composed of pure copper, so that only a small quantity of mercury is required to cover it. The result is that cleaning up is easy and rapid, no iron instrument being necessary, but rubber being always sufficient. These properties make it particularly valuable for custom mills, where it is desirable to catch as much as possible without mixing the amalgam obtained from two parcels of ore crushed in succession. On the other hand, as it holds little mercury, it cannot absorb much gold, and must be cleaned-up at frequent intervals.

The mercury on Muntz metal plates does not suffer so easily from "sickening" as that on copper plates; it has been suggested that this is due to the electrolytic action of the copper-zinc couple, which sets free nascent hydrogen, and so reduces the compounds of mercury and other metals which have been formed.

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* *Journ. Chem. Met. and Mng. Soc. of S.A.*, May, 1904, p. 399.

† See T. A. Rickard, *Stamp Milling of Gold Ores*, pp. 179-182.

It follows that Muntz metal plates are preferable for ores containing large amounts of heavy sulphides or arsenides. The greenish-yellow stains (called "verdigris" by millmen) which are formed on copper plates when grease and other impurities are present in the battery water, do not appear when Muntz metal is used, and such discolorations as occur on these plates can be better removed by dilute sulphuric acid than by potassium cyanide. At the Saxon Mill, New Zealand, the copper plates formerly required 7 lbs. of cyanide, costing 23s., per month to keep them in order, while the Muntz metal plates, by which they were replaced, could be kept clean by 5 lbs. of sulphuric acid per month, the cost being 3s. 4d. It is stated, however, that, in the treatment of highly acid ores, which have been weathered for some time so that they contain large quantities of soluble sulphates, or in cases where the battery water contains acids, copper plates are less affected than Muntz metal, over which a scum is rapidly formed. In the Thames Valley, N.Z., Muntz metal is preferred in spite of the extremely acid nature of the water and ore.

In dressing new Muntz metal plates the following method is adopted in New Zealand :—The surface of the plate is scoured with fine, clean sand; then it is rinsed with water, and washed with a dilute (1 to 6) solution of sulphuric acid. Mercury is then applied and rubbed in with a flannel mop until it wets the surface of the plate (*i.e.*, amalgamates with it) in one or more places, after which the mop is given a circular movement, passing through these spots, until the amalgamation of the surface spreads from them over the whole plate.

The discoloration of the Muntz metal plates is prevented by the weak electric current produced, as has been already stated. The same effect can, according to Aaron, be obtained when ordinary copper plates are in use, by placing them in contact with iron or some other metal which is positive to copper. Strips of iron bolted to the top and sides of the plate are said to be sufficient for the purpose, the copper being in that case unaffected by the acidity of the water, which causes oxidation and dissolution of the iron only. Janin's experience does not support these views.

Shaking Copper Plates.—A shaking copper plate has been recommended to be used either below or in place of the ordinary amalgamating tables, especially in cases where these do not appear to give good results. An ordinary fixed copper plate requires an inclination of from 1 to 2 inches per foot, in order to keep it clear of sand, when the plate is of the same width as the battery. If, however, the plate is subjected to a short rapid shake, the sand is kept from packing, and amalgamation is well performed with a grade of only ¼ to ½ inch per foot, or the am[illegible] of [illegible]led with the pulp may be greatly reduced and better contact thus obtained. For these plates, silver-plated copper is the material employed. They are affixed to a light wooden frame which is moved by a crank-shaft, revolving 180 to 200 times per minute, placed on one side, with a throw of 1 inch at right angles to the direction of the flow of the pulp. In some mills, a longitudinal shake is given to the plate instead of this side shake. The frame may be hung on rods from above, but is more conveniently supported on four short iron springs, forming rocking legs. The width of the tables is made as great as possible, while the length is of less importance, as, the thinner the current of pulp flowing over them, the better the chance of the gold particles coming in contact with the plates and being retained. These shaking plates were first used in Montana in 1878, and have since been employed at several Californian mills. It is advantageous to add to them an amalgam- and mercury-saver. A simple device for this purpose is to nail a strip of wood, half an inch thick, across the copper plate near the top, thus forming a shallow riffle, the angle of which is soon filled with sulphides and coarse sand, which are kept in agitation by the movement of the table. This is stated by W. M'Dermott and P. W. Duffield* to be the most effective contrivance yet devised for catching quicksilver and hard amalgam. If the inside copper plate should become hard by accident or neglect, chips of amalgam escaping through the screens are retained in this riffle, and, becoming spherical by rolling up and down under the effect of the shaking motion, increase in size just as a snowball does when rolled in snow.

In 1904, P. Carter† described the shaking plates used below the ordinary plates at the Ferreira Gold Mine. Each plate caught an average of 15·05 ozs. of fine gold per month. Each ordinary plate at the same time was catching nearly 400 ozs. per month. Two-thirds of the values caught on the shaking plates came from black sands. The plates were 11 feet long by 4 feet 6 inches wide. They were fixed on old Frue vanners, and had a fall of 4 inches in 12 feet.

Mercury Traps.—Another method of saving mercury and amalgam, which would otherwise be lost in the tailings, consists in the application of mercury wells or riffles. A mercury well consists of a shallow gutter filled with mercury, over the surface of which the pulp flows or through which it is forced to pass by suitable machinery. Attwood's amalgamator, formerly much used in California, was a machine of the latter class. Such wells or traps are usually placed between the successive plates, the pulp dropping from the end of a plate on to the surface of the mercury in the well, and then passing on to the next plate. The practice of placing a well between the screens and the amalgamating tables has been condemned, as it prevents proper supervision being kept over the feeding of mercury into the battery, over-feeding being difficult to detect under these circumstances.

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* *Gold Amalgamation*, London and New York, 1890, p. 16.

† *Journ. Chem. Met. and Mng. Soc. of S.A.*, May, 1904, p. 405.

Galvanic Action in Amalgamation.—In amalgamation in the mortar, on plates, or in pans, not only are free metals absorbed, but the dissolved salts, and, to a less extent, the insoluble compounds of the heavy metals, are reduced and amalgamated, chiefly by galvanic action. The copper of the plates, or the iron of the mortar or pan, constitutes the positive element, and all metals less oxidisable than this reacting metal are reduced by it, and are then amalgamated by the mercury. In this way iron reduces both lead and copper, although, if these are present in the form of undecomposed sulphides, this action will be very slight. Now, if lead is introduced into the amalgam, the latter becomes pasty, and is subjected to considerable losses, and copper has an equally harmful effect. It is for this reason that the arrastra is found to be better than the stamp battery or even the pan for certain plumbiferous ores. This action of iron is of course enormously increased if the ores are subjected to a chloridising roast before being amalgamated, as in the Reese River process for the treatment of auriferous silver ores.

In some mills, this galvanic action has been increased by the passage of a weak electric current through the charge by means of a dynamo. The amalgamated plates or the walls of the pan are connected with the negative pole, while the positive pole is formed of a plate of lead or iron dipping into the pulp. Under these conditions the mercury is still further protected from attack, and remains bright and lively, but the deposition of base metals in it is favoured, and the stronger the current the more this action is induced. Consequently, such methods are attended with the best results when dealing with ores containing little or no copper, lead, &c., since in these cases the strength of current can be increased, and the mercury kept clean, without any ill effects. The principle is made use of in Bazin's centrifugal amalgamator, Molloy's hydrogen amalgamator, and other similar machines.

Designolle Process of Amalgamation.*—In this process a solution of mercuric chloride is used. It was tried at the Haile Mine, South Carolina, the method being as follows:—Charges of 600 lbs. of roasted ore were placed in cast-iron barrels with 1,000 lbs. of cast-iron balls, or in pans. The barrel was partly filled with water, and 1 gallon was added of a solution containing 1·7 per cent. of mercuric chloride, the same amount of hydrochloric acid, and twice as much salt, if the ore contained less than 15 dwts. of gold per ton. This would be equivalent to about 10 parts of mercury to 1 part of gold. The barrel was rotated for twenty minutes and then discharged into a settler, and the suspended amalgam caught on copper plates. The mercury was supposed to be reduced by the iron thus—

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* This account is abridged from that given by Louis Janin, Junr., in *Mineral Industry in 1894*, p. 346.

$$HgCl_2 + Fe = FeCl_2 + Hg,$$

and the metallic mercury thus freed amalgamated with the gold. If no common salt was present, some mercurous chloride was formed according to the equation—

$$2HgCl_2 + Fe = Hg_2Cl_2 + FeCl_2,$$

and the subsequent reduction of the insoluble calomel by iron was not complete. Hydrochloric acid was supposed to hasten the amalgamation by setting up some electrolytic action.

The total cost of the process at the Haile Mine was said to be only 35 cents, or 1s 7½d., per ton, but it was abandoned when the percentage of iron in the material under treatment increased, owing to improved methods of concentration. Large quantities of oxide of iron were then amalgamated, and rendered the resulting mass harder to treat than the ore itself. By repeated washing, settling, and regrinding with fresh mercury, it could be partially purified, but not without a loss of gold. It is stated that 87 per cent. of the gold in the ore was extracted.

The Clean-up.—The amalgam, both on the inside and outside plates, does not accumulate evenly, but in ridges and knots which serve as nuclei for the collection of more. It is not advisable to allow the coating of amalgam to become very thick, since, although the plates catch better as the amalgam accumulates, losses may be experienced by scouring. The scraping of the outside plates usually takes from ten to fifteen minutes for each battery. The amalgam so obtained is ground with more mercury in a clean-up pan in order to soften it, the skimmings from the mercury wells, &c., being added to the charge. The inside plates are not scraped until the amalgam stands up in ridges on it; the operation may be necessary as often as twice a week, but it usually takes place twice a month, when a general clean-up is made. All amalgam, however obtained, unless it is already hard and dry, is usually at once separated from the excess of mercury contained in it by being squeezed in filter-bags, and the pasty residue alone passed to the clean-up pan or the retort.

In cleaning-up, the stamps are hung up, two batteries at a time, the screens, inside plates, and dies are all taken out and washed in tubs, and the "headings," or contents of the mortar, consisting of the pulp, mercury, sulphides, and pieces of iron and steel, amounting in all to a quantity sufficient to fill two or three buckets, are carefully scraped out and panned or fed into the mortar of one of the other batteries, which has not yet been cleaned up. In California, the headings from the last batteries are panned, the iron removed by a magnet, and the remainder ground with mercury in the clean-up pan. G. O. Smith states* that mortar-box sands, as well as the sands from mercury traps and from washing the screens, are very rich, and must be ground with mercury in a clean-up pan. Roskelley† considers that mortar-box sands from 10-dwt. rock would contain about 10 ozs. of gold to the ton after a three months' run. Amalgam is found adhering to the inside of the mortar and to the dies, and is carefully detached and added to the clean-up pan, and all the plates are well scraped with a sharp-edged piece of hard rubber or an iron tool, care being taken not to scratch them. Hard amalgam is removed by a chisel, care being taken not to lay bare the copper surface. Fresh mercury is then added, and brushed over the plates, which are finally smoothed with a soft brush. After the plates have been redressed, the batteries are restarted, and the next ones stopped and cleaned up. Three men can clean up forty stamps in from five to seven hours, ten stamps being thus idle for the whole of this time.

The amalgam obtained from the batteries, outside plates, mercury wells, or sluices, is rarely clean enough for immediate retorting; it is usually found to contain mixed with it grains of sand, pyrites, magnetite, and other minerals, together with fragments of iron and other foreign substances. The skimmings from mercury wells are still more impure. These materials must be purified by grinding with fresh mercury and washing before they can be passed to the retort. The scraps of iron consist of fragments of shoes, dies, shovels, picks, hammers, and drills: they are knocked about in the mortar until a quantity of gold and amalgam has been driven into their interstices. At the Jefferson Mill, Yuba County, California, about ½ ton of such scrap, picked out by hand or by a magnet, had accumulated in 1885. It was attacked by warm dilute sulphuric acid until the surface had all been dissolved off, and the residue was then well washed, and gold to the value of $3,000 thus recovered. The shoes, dies, &c., which were too large for this treatment, were boiled in water for half an hour, and then struck by a hammer, when the gold dropped out.‡ In small mills the dirty amalgam is ground in a mortar by hand with fresh mercury and hot water, until it is reduced to an even thin consistency, when the dirty water is poured off, and the mercury poured backwards and forwards from one clean porcelain basin to another until the pyrites, dirt, &c., have risen to the surface, when they are skimmed off. The skimmings obtained are put back into the mortar, and re-ground by themselves with fresh mercury. The clean mercury is then squeezed through canvas or wash-leather, when the greater part of the gold and silver contained in it, together with about one and a-half times its weight in mercury, remains in the bag, the rest of the mercury, with a small quantity of the precious metals dissolved in it, passing through. The mercury was formerly always squeezed by hand, but it is now sometimes squeezed by machinery, a ram working in a cylinder pressing out all surplus mercury from amalgam contained in a canvas bag. The ram is worked by compressed air or water.

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* *Journ. Chem. Met. and Mng. Soc. of S.A.*, June, 1904, p. 454.

† *Op. cit.*, Aug. 1904, p. 53.

‡ *Sixth Report Cal. State Min.*, 1886.

In large mills a clean-up revolving **barrel** is often employed to mix the amalgam. This is made of iron, and is similar in construction to chlorination barrels, but without the lead lining. At the Plumas Eureka Mill, the barrel is 3 feet in diameter and 4 feet long, and revolves twenty times a minute; the charge is 700 lbs. of amalgam and 20 lbs. of mercury, or more if the amalgam is very rich. A dozen or more iron balls or pieces of iron, such as worn-out battery shoe shanks, are put into the barrel, which is filled nearly up to the top with water, and then revolved for from six to twelve hours. The use of the iron is to help to mix the amalgam and mercury, but it causes some loss by flouring, and is omitted in Australian mills. The barrel is then opened and washed out with water, the tailings being run over amalgamated plates and through a mercury well or some other form of amalgam-saver, after which the amalgam is scooped out of the barrel and squeezed in wash-leather.

The **Clean-up Pan** is even more extensively used than the barrel. One of the oldest in use in the United States, the **Knox Pan,** is still a great favourite, especially for treating battery sands, skimmings, &c. It consists (Fig. 38) of an iron pan, 5 feet in diameter and 14 inches deep. Wooden or iron shoes are attached to the arms, *g*, which make from twelve to fourteen revolutions per minute. Iron shoes are considered better for brightening or polishing the particles of gold contained in the pyrites, and so rendering them fit for amalgamation. The charge for this pan is about 300 lbs. of impure amalgam, mercury, concentrates, skimmings, &c. The charge is made into a pulp with water and ground for three or four hours, after which more mercury is added, and mixing is carried on for a few hours longer, before the pulp is diluted, settled, and discharged. The tailings suspended in the water are usually passed over amalgamated plates, and are then often caught in settling pits, and either sold or subjected to further treatment on the mill, as they are frequently of high value. The mercury is squeezed through canvas, and the amalgam retorted.

At the Simmer and Jack Proprietary Mines, where the mill contains 280 stamps, there are five clean-up pans 36½ inches in diameter by 13½ inches deep.*

The position from which the greater part of the gold is obtained in a clean-up varies according to the ore and the method of treatment. In California from 50 to 80 per cent. of the gold saved is caught on the single plate inside the battery, the remainder being caught on the outside plates and the sluice plates, or being contained in the concentrates. In the Grass Valley, at the Original Empire and the North Star Mills, from 70 to 85 per cent. was caught inside the battery. The amalgam from the battery plates is usually richer than that from the outside plates, especially if the gold is coarse. The reason for this is that coarse gold, being easily amalgamable, is almost all caught on the inside plates, while fine gold, even if amalgamated in the battery, forms a more fluid amalgam which passes through the screens and is caught outside. Coarse gold forms a richer and stiffer amalgam than fine gold, for the reason already given on p. 15. At the two mills last named the value of the plate amalgam was $4.50 per oz., and that of the battery amalgam $8.50 per oz. According to Peplar† the amalgam caught on inside plates is of very low grade, on account of the large percentage

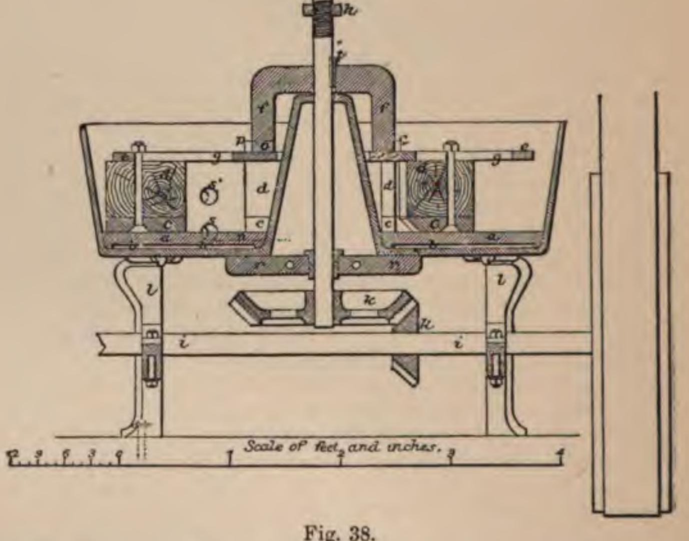

Fig. 38.

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* *Metallurgical Engineering on the Rand.* By John Yates. 1898, p. 47.

† *Journ. Chem. Met. and Mng. Soc. of S.A.*, May, 1904, p. 402.

of iron that amalgamates on them. Roskelley* found at the Robinson Deep Mine, where inside plates are not used, that 95 per cent. of the gold saved by amalgamation was caught within 3 feet of the top of the table.

About every three or six months the plates are scraped with a sharp iron chisel or palette knife, or scaled by chipping with a hammer and chisel. At the North Star Mill, California, the plates are immersed in boiling water so as to soften the amalgam before they are scraped. On the Rand the plates are generally steamed and scraped every two or three months, according to Denny. They are sometimes "sweated" in California by heating them over a wood fire before scraping them. At the Empire Mill, Grass Valley, California, the sweating of the outside and apron plates of four batteries produced bullion to the value of $19,000.† These plates had been down for eighteen months, and the ore which had been run over them averaged 18 dwts. of free gold per ton. After scaling and sweating, the plates may require replating. In course of time they are worn out, the copper becoming brittle and worn into holes, but they usually contain gold enough when discarded to pay for a new set.

Several methods are in use for recovering the gold from old plates. For example, they may be dissolved in nitric acid, when the gold is left nearly pure. A more economical method of detaching the gold, much used in Australia, is described by W. M'Cutcheon ‡ as follows:—The plates are placed on the hearth of a reverberatory furnace, or on a fire made with logs in the open air, and the mercury expelled at a gentle heat. If the temperature is too high, the gold sinks into the copper at once, and the copper must then be dissolved. After the mercury has been driven off, the plate appears to be more or less coated with gold on one side. This surface is treated with hydrochloric acid for eight or ten hours, and the plate is then replaced on the hearth and exposed to a dull red heat until well blackened. On plunging it into cold water, the gold now scales off, and is collected and freed from copper by boiling in nitric acid.

The method in use on the Rand for scaling plates before casting them aside is described by I. Roskelley as follows:—§ A mixture of ½ lb. sal-ammoniac, ½ lb. nitre, ½ lb. hydrochloric acid, and a pint of water is applied to the plate with a soft brush, and allowed to stand for about fifteen minutes. Afterwards the plate is heated over a good fire. When it has become quite black, which will be in about half an hour, it is dipped into a bath of water, when the scaling can be washed off.

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* *Op. cit.*, Feb. 1904, p. 287.

† *Eighth Report Cal. State Min.*, 1888, p. 714.

‡ Private communication.

§ *Loc. cit.*

Some millmen drive off the mercury before applying the above mixture; others do so afterwards. Should all the scaling not come off, the parts needing it are treated over again. The scalings are afterwards collected and mixed as follows:—1 part scalings, 1 part sulphur, 1 part borax, and 1 part sand. This is melted in a crucible and the gold extracted.

Old copper plates are often melted down and sold to refineries, where they are useful for mixing with gold and silver bullion in making-up the alloy for parting.

Retorting.—The solid amalgam, which is retained in the canvas or wash-leather filters, usually contains from 30 to 45 per cent. of gold and silver, according to the state of division of the gold present in the ore, and also to the degree of care exercised in squeezing out the excess of mercury. For separating the gold from the mercury there are two kinds of retorts in general use—the *pot-shaped* retort, which is sometimes cast with trunnions to swing on supports, in small mills; and the cylindrical retort, shown in Fig. 39, in larger mills. The pasty amalgam is rolled up into balls or kneaded into cakes, and squeezed into the pot-shaped retort, and often rammed down with a bolt-head, although this course is deprecated by some managers, who prefer to leave the amalgam as spongy and open in texture as possible, believing that a uniform product is thus obtained more rapidly at a lower temperature and without so

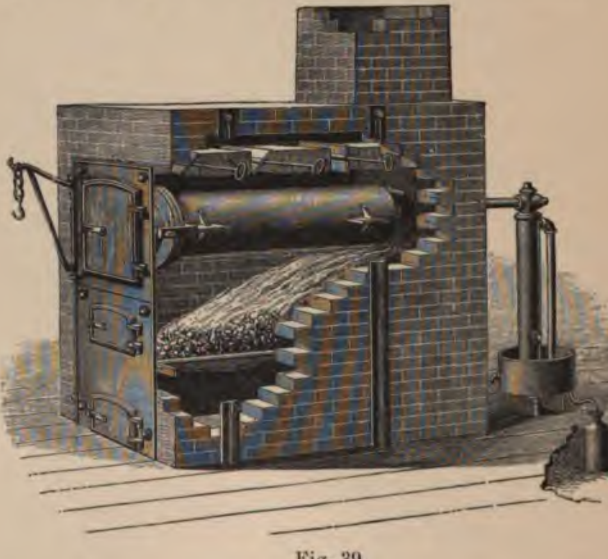

Fig. 39.

much loss. In the horizontal retort the amalgam is placed in iron trays divided into compartments by partitions. In either case, the retorted metal is prevented from adhering to the iron, either by laying it on three or four thicknesses of paper, the ashes of which remain beneath the amalgam, or by covering the iron trays with a coating of whitening. The mercury is condensed in cooling tubes passed through water; the loss through volatilisation is usually very small, and may be taken as being about one grain of gold per pound of mercury.

The charge is heated slowly until the boiling point of mercury is reached, when the fire is checked, and the retort kept at an even temperature for three or four hours, or until the bulk of the mercury has been driven off. The retort is then raised gradually to a bright red heat to expel the remainder; after cooling, it is opened, the trays withdrawn, the retorted metal loosened by a chisel, if necessary, and turned out on a table.

In retorting amalgam containing considerable quantities of base materials, there is a danger of the vent being choked up by condensation of solid material. The retort should be so arranged that a rod can be passed through the condensing pipe so as to clear it of obstructions, if necessary. The front of the retort is luted on carefully with chalk or wood-ashes and salt, and firmly clamped so as to be quite tight, otherwise a loss of mercury is incurred. In all retorts the lid is turned and ground so as to fit on perfectly. The condensing pipe should not have an open end dipping freely into water, as in that case a sudden cooling of the retort would cause the water to be sucked in, and an explosion would occur. The open end of the pipe may be enclosed in a bag of sacking or rubber immersed in water, or the pipe may be continued by a piece of sacking which dips under water.

The pot-shaped retort requires no brick fittings, and can be heated over an assay furnace or forge fire, or in a fire built on the ground, when it is placed on a tripod stand. In the latter case the fire is lit at the top and burns slowly downwards. The pot-shaped retort is not filled to more than two-thirds its capacity, and must be heated very gradually at first.

The retorted metal is porous and from 500 to 950 fine in gold, the remainder being in general chiefly silver, with base metals and sulphides in smaller quantities. The gold is melted in crucibles with sand, carbonate of soda, and borax, and suffers a further loss in weight, due to the slagging off of oxides, earthy impurities, &c., and to the volatilisation of a small quantity of mercury, which is obstinately retained until the melting takes place. This amount need not be more than from 0·5 to 1 part per 1,000 of the retorted metal, if the retorting has been carefully performed and finished at a high temperature.

Richards states[*] that if the amount of mercury is to be reduced below 1 or 1½ per cent. in retorting, a white heat must be used, by which the retorts are damaged and soon worn-out. Additions of nitre, corrosive sublimate, &c., are not to be recommended. The melting of bullion is dealt with fully in Chapter xviii. The loss of weight in melting is given by Richards[†] as 1½ per cent. at the Homestake Mill and 7 per cent. in the Caledonia Mill. Part of this loss consists of gold and silver retained by the slag which is either remelted or passed to the stamp mill or clean-up barrel.

Loss of Mercury.—The loss of mercury is due to two causes, "flouring," or minute mechanical subdivision, due to excessive stamping or grinding, and "sickening," or extreme subdivision caused by chemical means.[‡] In the latter case, a coating of some impurity is formed over the minute globules of mercury, which are thereby prevented from coalescing, from taking up gold and silver, or from being caught by the plates and wells, as the coating prevents all contact between the mercury and other bodies. The impurity may be an oxide, sulphate, sulphide, or arsenide of some base metal, either originally present in the mercury, or taken up from the ore by it; occasionally the mercury itself may be partly converted into a sulphate or other salt, although this latter condition is not common. The employment of mercury, which contains no base metals dissolved in it, will reduce the loss due to sickening, but such pure mercury is not always obtainable (except by very careful distillation, in which the first and last portions condensed are rejected), and it soon takes up fresh impurities when used with sulphuretted ores. The base metals usually present in mercury are rapidly oxidised in the air, especially in contact with water; the oxidation is made much more rapid by the presence of any acid in the water, and this acidity (due to the presence of acid sulphates from decomposing pyrites) is rarely quite absent from battery and mine waters. The metallic oxides thus formed are not soluble in mercury, and they float on its surface in the form of little black scales, which soon form a coating. Impure mercury, when used to amalgamate the plates, causes their discoloration by oxidation of the dissolved metals. One of the impurities in mercury most to be feared is lead, as the amalgam of this metal tends to separate out of the bath of mercury in which it is dissolved. According to Prof. J. Cosmo Newbery, it rises to the surface by degrees, taking with it any gold amalgam that may have been formed, and floats as a frothy scum, coating the mercury and preventing any further action by it, whilst it is readily powdered and carried away in suspension by a current of water flowing over it, so that the gold contained in it is lost.

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[*] *Ore Dressing*, vol. ii., p. 784.

[†] *Op. cit.*, p. 784.

[‡] Richards points out (*Ore Dressing*, p. 751) that there is no advantage in trying to draw a distinction between these two terms, which was apparently first done by T. A. Rickard. Richards holds the view that the globules are in all cases prevented from re-uniting by a film of some foreign substance.

Sickening of the mercury is also promoted by base minerals present in the ore. Most gangues, except heavy spar, hydrous silicates, &c., have no action on the quicksilver; even clean cubical iron pyrites, and other iron and copper pyrites, if they are undecomposed, are harmless, although the materials last named cause sickening when partly decomposed. The other sulphides are all more or less harmful, their action being, however, much less energetic than the compounds of arsenic and antimony. J. Cosmo Newbery conducted a number of experiments in Australia many years ago, to determine the action of some of the base metals on mercury, and found that compounds of arsenic and antimony are particularly harmful, and that if gold containing metallic arsenic is amalgamated, the resulting amalgam is black and powdery, and floats on the mercury, being coated by black metallic arsenic, which separates out and refuses to unite with mercury. Arsenical pyrites seems to act in the same way as metallic arsenic, a large amount of black sickened mercury being produced by it, the action being especially energetic if the pyrites is partly decomposed. The black coating is, in this case, a mixture of pyrites, arsenic, and mercury, in a very finely divided state. Sodium amalgam acts beneficially when arsenic is causing loss of mercury.

Sulphide of antimony breaks the mercury into black powder even more quickly than arsenic, some sulphide of mercury being formed if there is any trituration, whilst the antimony forms an amalgam. The action of sodium amalgam on this mixture is of no avail, as sodium sulphide is formed, more antimony amalgam produced, and sulphuretted hydrogen set free, the results on the amalgamation of the gold being very disastrous. Bismuth sulphide acts similarly, but with less rapidity.

Floured mercury is perfectly white in appearance, like flour, sickened mercury, as already stated, being blackish. If this floured mercury is examined with a lens, it is seen to consist of a number of minute particles—many of them microscopic—each of which is perfectly bright and pure, shining like a mirror. They are prevented from coming into contact and coalescing by being surrounded by films either of air or of some transparent foreign substance. Floured mercury is readily carried away and lost in the tailings, but if passed through and agitated with a large body of clean mercury much of it is at once absorbed in the mass. The loss through flouring is experienced in the milling both of refractory and free-milling ores. The effects of grease and also of talc, serpentine, clay, and other hydrous silicates in subdividing mercury are doubtless due to mechanical action only.

In California the total loss of mercury varies from $\frac{1}{5}$ to 1 oz. of mercury per ton of ore crushed, the mean being about $\frac{1}{2}$ oz. per ton. Most of the mercury is lost as such and not in the form of amalgam, as is proved by the fact that where the largest proportion of mercury is fed into the battery the greatest loss takes place but the highest percentage of gold is recovered. Thus, at the North Star and Empire Mills the greatest loss in the State occurs, 1 oz. of mercury being lost per ton, but over 90 per cent. of the gold is extracted. In the Blackhawk Mills, Colorado, where base ores are crushed, containing from 12 to 20 per cent. of pyrites, the loss of mercury is from $\frac{1}{5}$ to $\frac{1}{2}$ oz. of mercury per ton. The mills in which the greatest loss of mercury occurs have the deepest discharge, the ore and mercury being in these cases pounded together for a greater length of time before being ejected from the mortar, so that more flouring takes place. At the Ferreira Deep Mill the loss of mercury is given by D. J. Peplar* as 0·35 oz. per oz. of gold and 0·16 oz. per ton of ore crushed.

Many suggestions have been made at various times for the reduction of the loss of mercury. The use of certain chemicals in keeping it clean and lively and in neutralising the bad effect of base minerals has already been noticed (see p. 130).

Properties and Purification of Mercury.—Pure mercury is unaffected by the air at ordinary temperatures, but is slowly oxidised if heated to about 350° C. It is not acted on by hydrochloric acid, and is almost unaffected by dilute sulphuric acid, but with hot concentrated sulphuric acid it forms $HgSO_4$. Mercury is dissolved even by cold dilute nitric acid, and is rapidly dissolved by hot nitric acid. Pure mercury will roll down an inclined surface without forming a pronounced "tail" and without leaving any streak behind it. If a blackish film is left behind, the mercury requires purification.

When agitated with oil, fats, turpentine, many organic substances, sulphur, &c., mercury is split up into minute globules, not easily re-united. This is known as the "flouring" of mercury. Vegetable or animal oils cause more flouring than mineral oils. Coalescence of floured mercury is effected by the action of certain reducing agents, such as water and sodium, the passage of an electric current, or with some loss by the action of nitric acid.

Amalgams.—Mercury forms amalgams directly with gold, silver (more readily if heated), copper, lead, zinc, and bismuth. Amalgams of tin and cadmium are formed directly with great ease. Mercury unites with antimony and arsenic only if heated. Antimony gradually separates from its amalgam as a black powder. Iron amalgam is formed directly only if the iron is finely divided. When iron amalgam is retorted, pyrophoric iron is formed, yielding a somewhat troublesome mixture with gold. Amalgams of nickel, cobalt, manganese, chromium, aluminium, and platinum are not formed directly, but are formed indirectly by electrolysis of their salts with mercury as the negative pole. Amalgams of sodium and potassium are formed directly with the aid of heat. The amalgam $Hg_{12}Na_2$ is solid, containing about 2 per cent. of sodium.

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* *Journ. Chem. Met and Mng. Soc. of S.A.*, May, 1904, p. 403.

Purification of Mercury.—It is very important to use pure mercury in ordinary amalgamation processes, so as to reduce the losses as far as possible. The purification may be effected by distillation with lime and iron filings. The iron filings decompose sulphides and prevent bumping. An addition of charcoal powder is mentioned by Richards, its use being to prevent the formation of volatile oxides. Lead and zinc pass over in part with the mercury. Zinc, tin, copper, and iron may be removed by shaking with dilute hydrochloric acid. Floating impurities are removed by running the mercury through a glass funnel, regulating the discharge by a finger placed over the stem-hole. If mercury is covered by dilute nitric acid (one part of acid to three parts of water) it is gradually purified, especially if stirred occasionally. Mercurous nitrate is formed and acts on the base metals. A more rapid way of removing base metals is to pass a stream of air through mercury covered with dilute nitric or sulphuric acid. The base metals are rapidly oxidised by the air and dissolved by the nitric acid. This method has been found useful by T. C. Cloud.

Loss of Gold.—The losses of gold in amalgamation may be ranged under the following heads:—

1. Loss of free gold contained in amalgam, due to flouring of mercury, scouring of plates, &c. This has been dealt with above under the heads "Treatment of the Plates" and "Loss of Mercury."

2. Loss of gold which "floats" in water and is carried away with the slimes. See section on "Float Gold."

The remedy for the losses due to the above two causes is the use of drops and mercury traps.

3. Loss of gold which is not in a condition to be directly amalgamated.

The latter heading may be subdivided into three, viz.:—

(*a*) Loss of gold contained in sulphides, tellurides, &c. See "Gold in Pyrites" and the treatment of tellurides described in Chapter xvi.

(*b*) Loss of free gold, which is prevented from being amalgamated by being coated with a film of some mineral ("rusty" gold), or with grease. See the section on "Rusty Gold."

(*c*) Loss of "free" gold imbedded in particles of rock. The remedy is finer crushing in the battery or in re-grinding machines.

In the preparation of gold ores for amalgamation every care must be taken that the course best suited to each particular case is being pursued. In some instances, which are not common, the whole of the gold may be present in a form in which it can be directly amalgamated. In general, however, the gold is present in two or more forms, one capable and the others not capable of amalgamation. In such cases there is no reason to be dissatisfied with the action of an amalgamating machine if it extracts a high percentage of the free gold, even though the total extraction obtained by it is comparatively low.

The following scheme of examining tailings with a view to determine the causes and amount of loss is given by M'Dermott & Duffield*:—

Small samples are taken at intervals from the waste outflow of the mill, until a bucketful is collected; this is allowed to settle for several hours, the clear water is decanted, preferably through a filter, and the remainder evaporated to dryness. Care must be taken to avoid spilling anything out of the vessels containing the samples. The sample having been well mixed, portions are treated as follows:—

A. One part is panned and examined for free gold, amalgam, and quicksilver. If these are present, it is probably the fault of the millman, and nothing further need be done until this state of things is remedied.

B. The tailings are sized by screening, and the coarse, medium, and fine materials (the latter consisting, say, of that portion which passes a 100-mesh screen) are weighed and assayed separately, the coarser portions being reground and panned to find whether their values are in free gold or in sulphides.

C. The sulphides are separated from the tailings on a vanning shovel or batea, and are weighed and assayed. It may thus be determined whether they are worth saving, and the size of the mesh used for the screens will depend largely on this, and on the nature of the sulphides, which will in many cases be badly slimed and difficult to catch if the ore is finely crushed.

D. The loss due to fine or "float" gold may be determined by assaying the slimes after the sulphides have been carefully removed by concentration. This requires much skill and patience, but can in almost all cases be successfully accomplished by the vanning shovel. The concentrates may be examined under the microscope for fine specks of gold, but these and the fine sulphides can be recovered by concentration on suitable machinery. The assay value of the tailings from the vanning shovel will give some idea of the amount of float gold which is being lost. It will usually be found to be smaller than may be expected. If it is large, the use of some system of amalgamation more perfect than that by copper plates (such as pan-amalgamation) or of a method of smelting, or of a wet method, may be considered, if the advantages appear sufficient to pay for the presumably increased cost.

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* *Gold Amalgamation*, London and New York, 1890, p. 7.

"Float" Gold.—The loss of finely-divided or "float" gold, particularly when it cannot be checked by the use of swinging plates, or of drops between the amalgamated plates, is often another name for the loss of slimed sulphides. Many examples have been adduced of the large percentage of the gold in the ores crushed in particular mills, which has been carried away suspended in water in a form not easily recoverable by settling. In the majority of these cases, however, no attempt seems to have been made to distinguish between the values contained in slimed sulphides and those existing as particles of free gold. Where this is not done there are no grounds for the assumption that any free gold is escaping at all. Thus G. M'Dougal, of Grass Valley, California, found* that a gallon of water in a stream, $\frac{3}{4}$ mile below two mills, contained on an average 1·18 cents worth of gold. He called this "float" gold, but did not try to find out its physical condition, and it was very likely contained in sulphides. Again at the Spring Gully Mine, in Queensland, the tailings from the battery, if settled in the ordinary way by running off the water, were found to contain 7 dwts. of gold per ton, but if carefully filtered, assayed 15 dwts. All such examples prove only that the slimes are rich, not that "float" gold is being lost, and although it is of course likely that some finely-divided gold is carried away in suspension in water during the treatment of many ores, nevertheless, if sufficient care were taken in ascertaining this loss, it would probably prove to be less than is generally believed.

"Rusty" Gold.—The appearance in the tailings of free gold, which is not especially finely divided, but, nevertheless, is not in a condition to be amalgamated, may be regarded as a rare occurrence, but deserves some consideration. Amalgamation is in these cases prevented by the existence of a thin film of some neutral substance over the surface of the gold. The film may be so thin as to be transparent, but it is enough to prevent contact between the gold and the mercury. The disastrous effect of a film of grease covering gold particles has already been remarked upon. It is said to have been a fruitful source of loss in the treatment of certain ores in the Transvaal that they were impregnated with mineral oil. The effects of grease may be combated by the use of chemicals (caustic alkalies, potassium cyanide, &c.), but it is, of course, better to use every precaution to avoid the introduction into the pulp of candle grease from the mine or of oil from bearings, guides, &c., or contained in steam from the boiler. Losses in amalgamation are also caused by the greasy substances contained in some ores, such as the powdered hydrated silicates of magnesia and of alumina, which cause frothing, and coat the gold with a slime which prevents the action of the mercury.

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* *Mines West of the Rocky Mountains*, by R. W. Raymond.

Others films are formed of oxide of iron, compounds of sulphur, arsenic, &c., or of silica. Some years ago J. Hankey, of San Francisco, had a collection of particles of native gold which appeared as bright and lustrous as usual, but were coated by thin translucent films of red oxide of iron. These particles of "rusty" gold could not be wetted by mercury, but, if a piece were snipped off one end, the mercury seized on the fractured surface at once. Such gold seems to be rare in nature.

In 1867, William Skey, of the Geological Survey of New Zealand, after a series of experiments on the ores and tailings of the Thames Valley, came to the conclusion that the bright gold particles which refused to amalgamate were always coated by some compound of sulphur. He found that gold takes up sulphur from sulphides of ammonium or sodium, or from sulphuretted hydrogen, when brought in contact with their solutions, and that after this the gold refuses to amalgamate. He supposed that these compounds of sulphur were often formed by the action of acidulated water on the minerals in ores, and that consequently "a large area of the natural surfaces of native gold is covered with a thin film of auriferous sulphide, and that the greater part of the gold which escapes amalgamation at the battery consists of this sulphurised gold."

Gold in Pyrites.—In the preceding pages no account has been taken of the loss of gold which is contained in pyrites, as it has been assumed that these are saved by concentration if they are valuable, and this subject is dealt with in Chapter ix. Nevertheless, as this gold comes under the head of non-amalgamable gold, its physical state and the causes of its disinclination to unite with mercury may conveniently be considered here. In general, pyrites yields only a small proportion of its gold contents if it is run over the amalgamated plates, and if it is ground very fine in a pan with mercury the percentage extraction is better. Among the old processes used for the amalgamation of the gold in pyrites may be mentioned the treatment in revolving wooden barrels with mercury, as practised at the St. John del Rey Mine, and the practice of leaving the pyrites to be decomposed by weathering before grinding it with mercury. This method of oxidation seems to be decidedly inferior to the alternative plan of roasting the sulphides, by which the oxidation is rendered more complete and the particles of gold agglomerated to some extent. However, the amalgamation of pyrites, even when roasted, is far from perfect, part of the gold still remaining in a condition unfit for extraction in this way. The ores, which have been met with in various parts of the world, consisting mainly of limonite or hydrated oxide of iron, and in most cases believed to be the result of decomposition of pyritic ores by atmospheric agencies, are also extremely refractory, causing the mercury to sicken rapidly, and yielding only about the same percentage of gold as can be obtained from unoxidised pyrites.

The most celebrated case of this kind is that of the surface ore at Mount Morgan, in Queensland, which was an ironstone gossan consisting of siliceous brown iron ore, derived according to one view from the decomposition of pyrites. Although the gold appeared to be free, it could not be amalgamated, yielding only about 30 per cent. when crushed in batteries and subjected to prolonged grinding in pans with mercury. When the ore was dehydrated by roasting in reverberatory furnaces the extremely fine particles of gold were agglomerated, and between 80 and 90 per cent. could then be extracted by amalgamation, the remainder being presumably coated by oxides of iron. The richness of the ore, however, made even this result unsatisfactory, and a process of chlorination was adopted in practice. A similar case was noticed by Mactear* in South America, where a limonite ore which only yielded from 35 to 40 per cent. of its gold when treated in Huntington pans, was made to yield between 85 and 90 per cent. by merely subjecting it to a dehydrating calcination before amalgamating it. Louis Janin, Jr., mentions another case † in the ores of the Southern Cross Mine, Deer Lodge County, Montana, which consist of limonite derived from the alteration of pyrites. In panning large samples, only one or two specks of gold could be seen, although the ore contained from 1 to 2 ounces per ton. This ore yielded only about 40 per cent. on being amalgamated, but over 90 per cent. was dissolved out by leaching the raw ore with cyanide of potassium, and similar results were obtained by chlorination. Here the ore was thoroughly decomposed, but yet the gold would not amalgamate to a much greater extent than if it were still contained in the original pyrites, whilst the chemicals at once dissolved it.

In seeking to explain the behaviour of gold in pyrites, various theories have been propounded. According to one of these, the gold is supposed to exist in the pyrites in the form of sulphide combined with sulphides of iron, silver, copper, &c., and the refusal of the gold to amalgamate is explained in this way, auric sulphide not being acted on by mercury. Some observers have endeavoured to dissolve gold out of pyrites by the action of alkaline sulphides, and when, after many attempts, this was at length successfully accomplished, it was put forward as additional evidence that the gold must have been in the state of sulphide, although metallic gold is known to be soluble in these menstrua.

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* *Mining Journal*, Jan. 24, 1893, p. 70.

† *Mineral Industry for 1892*, p. 249.

The balance of evidence, however, seems to be in favour of the theory that gold exists in pyrites in the metallic state. Although the metal is generally invisible in undecomposed crystals of pyrites, it becomes visible when such crystals are oxidised either by air and water in nature, or by means of nitric acid, or by being roasted or subjected to deflagration with nitre. As a result of such decomposition, particles of bright lustrous gold, angular and ragged in shape, but of considerable size, often become apparent. These particles may be separated from the oxides of iron by washing, and the use of nitric acid, followed by panning, is frequently resorted to in order to detect gold in pyrites. Moreover, although usually invisible, gold can sometimes be seen in unroasted pyrites. As long ago as the year 1874, Richard Daintree and Latta found specimens of cubical pyrites,* in which gold could be seen under a microscope gilding the cleavage planes of the crystals. Again, G. Melville-Attwood, on examining crystals of auriferous pyrites from California in 1881,† found that the faces of the crystals were gilded in some places, and that here and there little specks or drops of gold occurred, partially imbedded in the pyrites. These films were too thin to be detected by an ordinary lens, so that it did not seem surprising that such impalpable material could not be taken up by mercury. Louis Janin, Jr., more recently‡ found crystals of pyrites in a porphyritic gangue from the Republic of Colombia, which had gold in small globules on their surfaces. Lastly, it has long been known that crystals of pyrites are often found adhering to an amalgamated plate, the particles of gold on their surfaces having been amalgamated. It seems likely, in view of all these facts, that most if not all of the gold is in the metallic state, and its occasional refusal to amalgamate is not very surprising, when it is remembered how completely a thin coating of certain sulphurised compounds prevents amalgamation, and how readily sulphuretted hydrogen would be evolved from decomposing pyrites. Some authorities have contended that the metallic gold is disseminated mechanically through the mass of pyrites, but the action of potassium cyanide, in dissolving the whole of the gold out of comparatively coarsely crushed pyrites, seems to point to the correctness of the view that the interior of the crystal is not auriferous, the deposition of the gold being superficial, so that the enrichment of the pyrites is confined to its crystalline faces, and possibly, but not probably, to its cleavage planes. Strong evidence of the richness of the outside of the crystals of pyrites in the

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[illegible] *Soc. of New South Wales*, 1874; paper on "Iron Pyrites."

[illegible] *ıls in the United States*, 1881, p. 604.

[illegible] *stry*, 18 [illegible]

Banket ore of the Transvaal is given by Caldecott in his papers on the treatment of ore in the Transvaal.*

The following details † of a microscopical examination by Prof. Morton, of the condition in which pyrites is left after being leached with cyanide, confirms this view to some extent:—

"Upon the ordinary auriferous sulphide of iron, or arsenical pyrites, the solution of potassium cyanide acts readily, not by dissolving the sulphuret, but by attacking the gold upon its exposed edges, and eating its way into the cubes by a slow advance, dissolving out the gold as it goes. An examination with the microscope of the pyrites after the gold has been removed, suggests the method of the operation. A sample of very rich pyrites from a mine north of Redding, was treated with a weak solution, containing less than two-tenths of 1 per cent. of cyanide, for 168 hours; the assay showed a complete extraction of the gold; as the sulphurets showed no change in their appearance to the naked eye, some of them were placed under the microscope.

"There is no change visible in the form of the crystals as a whole; along the fractured faces the mispickel looks clean and unaltered, showing the silvery-white colour and intense refraction of the arseno-pyrite. Upon the faces of the crystals appear dark lines, short, and parallel to each other. In places they are crowded close together; in other parts they are at considerable distances, but always in parallel lines. The lines vary in length, being from four or five to over a hundred times their width; the lines are very irregular and often broken. These lines are fissures in the pyrites, and extend so deep into it that the microscope does not reveal their depth. By using the higher powers the walls of one of the fissures were seen to be completely honeycombed, looking somewhat like two empty honeycombs set opposite each other; evidently the mineral removed was crystallised along its contact walls at least. As the raw or untreated pyrites does not show any such fissuring, but, upon the contrary, shows a surface marked only by striation lines common to pyrites, I assume that the fissuring in the treated sample is caused by the solution acting upon some soluble mineral, probably gold, arranged in plates, occurring in groups, but which, by its colour and isomorphism and the extreme tenuity of its lines, is undistinguishable from the mass of pyrites enclosing it."

Discussion of Stamp-milling.—The stamp battery must be regarded from two different points of view—viz., (*a*) as a crushing machine, (*b*) as an amalgamating machine, and it should be remembered that the modifications designed to make it a more efficient crusher often reduce its power as an amalgamator, and *vice versâ*.

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* *Journ. Chem. Met. and Mng. Soc. of S.A.*, December, 1903, p. 237. *Trans. Inst. Mng. and Met.*, October, 1904.

† *Mineral Industry*, 1892, p. 249.

Stamps were originally designed as crushing machines, and the tendency is just now towards a return to this view of its proper functions. A practice which now seems to be growing in popularity consists briefly in crushing the ore and effecting its discharge from the battery as rapidly as possible. With this object in view heavy stamps are used, running very fast, with a moderate drop; the screens are coarse, the screen area is large and placed as low down as possible, and the mortar is made narrow, with nearly vertical sides. These arrangements all increase the output of the battery. Amalgamated copper plates are either not placed inside the mortar, or there is a plate on the discharge side only. In the extreme case, no mercury is fed into the battery, and the coarsely-crushed ore passes over amalgamated plates and is then re-ground in tube mills, and either again run over the plates or passed at once to the cyanide plant. Under these conditions, the output is very large, and may even be as high as 8 to 12 tons of ore per stamp per day. The percentage of gold extracted by amalgamation is low.

When the stamp battery is used as an amalgamator, the depth of discharge (height of screen above dies) is great, amounting even to 12 inches or more, the screens are finer, the supply of water is reduced, and all efforts are directed to fine crushing in the battery. Mercury is fed into the mortar, and amalgamated plates are put inside the mortar on both feed and discharge sides. The object of these arrangements is to keep the ore in the mortar for a long time, so as to increase the chance of catching the gold on the inside plates. The duty of the stamps is, of course, greatly diminished, and may be as little as one ton of ore per stamp per day. Under such circumstances a high percentage of gold is amalgamated. Between these two extremes there are many gradations.

It is probable that no two ores can be treated to the best advantage under exactly the same conditions. A millman experienced in the treatment of the ores of one district may be quite at fault when attempting to amalgamate an ore unlike those to which he has been accustomed. A silver mill, in particular, has been pronounced to be the worst possible school for a gold amalgamator, whose work must be closer in proportion as his amalgam is richer than that obtained from silver ores.

It is obvious that the stamps and screens must be such as are calculated to produce the largest possible output, without rendering the pulp unsuitable for the processes of amalgamation or of concentration, or both, which are to follow. The ideal crushing would be to "crack the nut and leave the kernel entire," or in other words, to liberate the particles of gold without breaking them. This suggestion, however, is not a helpful one, inasmuch as it has been found in many cases that amalgamable gold remains locked up in particles of ore of all sizes except the very finest slimes. The tendency is now in the direction of finer crushing, if not in the stamp battery, then in re-grinding machines (see below, Chap. xi.).

The subject of delivery is closely connected with that of crushing and must be considered at the same time. The screens are not usually placed quite close to the level of the dies in the mortar on account of the rapid wear caused by the violent projection of pulp against them when in that position. Their height above the dies is varied according to the ore, the delivery being slower in proportion to this depth of discharge when it exceeds 2 or 3 inches or, according to Peplar, 5 inches. The banking of sand against the screens checks discharge through their lower part if the depth of discharge is small.

The best size for the mesh of the screens must be determined by direct experiment. It has often been contended that, as the crushing must be fine enough to liberate the particles of free gold from their matrix, therefore the size of the screen mesh depends on the state of division of the precious metal in the ore. Even if this be so, however, it does not follow that the apertures in the screen need be small. When coarse screens are used, it is found that, in the course of the crushing, much of the ore has been reduced to a comparatively fine state of division, and usually this portion is found after amalgamation to contain but little gold; from this, the coarser material, in which the gold is still locked up, may be separated by sizing in "pointed boxes" (see p. 175) and reground in an amalgamating mill or pan. If, on the other hand, the slimes are found to be as rich as the coarse sand, it is obvious that no finer crushing is required, as the output would be thereby diminished without any corresponding increase in the yield per ton. If the slimes, after separation of the sulphides, are found to contain more free gold than the coarse sand does, this fact points to the conclusion that a coarser screen might be used without detriment, and experiments in this direction should be made, and the limit of economy thus found by trial.

The evils of overstamping, due to slowness of discharge, have been often dwelt on, and probably are frequently exaggerated. It is true that the excessive production of slime thus caused may sometimes be disadvantageous, but, setting this aside, it has been frequently asserted that particles of gold are reduced in size by overstamping, so that they will float off in suspension in water, whilst, even if not reduced in size, they are hammered and flattened so as to be rendered incapable of amalgamation. "Float" gold has been considered above, p. 147, and as regards hammered gold, it does not seem to be beyond doubt that flattening and hardening alone will prevent gold from being amalgamated. Prof. T. Egleston has described a number of experiments * which tend to show that amalgamation is retarded by this treatment of gold, but, on attempting to repeat these experiments at the Royal Mint, the author could not obtain similar results to those of Professor Egleston. Pieces of pure gold when subjected to repeated blows with a clean 7-lb. hammer on a clean anvil occasionally showed a disinclination to amalgamate, but if these pieces were washed with dilute ammonia, so as to remove any grease that might be adhering to them, they were instantly wetted by mercury and were dissolved by it at about the same rate as clean annealed gold. It thus appeared that, in these cases at least, grease formed a more potent preventive of amalgamation than the hardness of the gold. Moreover, gold-leaf which has been subjected to an extended course of hammering is readily amalgamable. It seems probable that the supposed difficulty of amalgamating "hammered" gold has no real existence.

There is still much difference of opinion as to the desirability of attempting to amalgamate the gold inside the battery. It is considered by some millmen that the addition of mercury to the mortar is a mistake, and that no copper plates should be put inside. In spite of numerous practical illustrations of the soundness of the contrary view, they hold that no machine can be successful at once as a crusher and an amalgamator. The practice of adding mercury to the mortar when no inside plates are used is certainly not now much favoured, although it is still adhered to in Ballarat and some other districts in Australia. In treating rich ores, however, when the gold is coarse and of high standard, there does not seem to be any valid objection to be raised against catching the gold on inside plates in a concentrated form, instead of letting it all go to the outside. In this case mercury must be added to the mortar, and this is probably not so important a cause of loss of mercury by flouring and sickening as is often assumed. If a decomposing ore is mixed with lime beforehand, and the acidity of the water used is corrected, the conditions do not appear to be favourable to the production of salts injurious to the mercury, and the latter when charged in is probably almost instantly washed through the screens or else dashed against and retained by the plates. The mercury does not remain on the die, subjected to repeated blows, which would no doubt cause much flouring. There is also the point of view that the distribution of mercury through the ore favours amalgamation, and that amalgam is more easily caught on stationary or shaking plates than very finely-divided gold. This last point remains a matter of some doubt. It is often denied on the ground that amalgam is of less density than gold.

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* *Metallurgy of Gold, Silver, and Mercury in the United States*, 1887, vol. ii., p. 586.

Finely-divided gold, however, particularly if it is of low standard, containing much silver, requires more mercury for its amalgamation than coarse gold, and in this case it is difficult to keep the plates in good order, so that it is usually advantageous to save the extra trouble and labour, caused by looking after and cleaning-up inside plates, by putting all the plates outside. This has been the experience in a number of mills, including that of the Montana Company, where the inside plates have been entirely discarded.

Amalgamation outside the battery has also been the subject of much discussion and some careful investigation. Numerous amalgamating machines have been patented, the inventors in every case praising their own contrivances and decrying the copper plate, but the latter has not as yet been superseded, and its principle is applied to almost all its more promising rivals.

To secure successful amalgamation it is necessary that the particles of gold should be brought into absolute contact with the mercury. This contact is obtained in one of three ways, viz. :—

1. The mercury and ore are ground together in pans, arrastras and similar machines, contact being secured by pressing the gold and mercury together.

2. The ore is allowed to flow over or even through a bath of liquid mercury, or the endeavour is made to ensure contact by letting the pulp fall from a height upon the mercury.

3. The ore is allowed to flow over either stationary or shaking amalgamated copper plates, drops being sometimes introduced between the plates to break up the pulp and to assist in catching the amalgam and "float" gold.

The first method is undoubtedly the best for ensuring contact, but the operation is tedious and, in most cases, unnecessary. As to the relative merits of the two latter methods, the great majority of metallurgists are advocates of the superiority of the plates. They point out that in a mercury bath, in spite of the first impression to the contrary felt by every one on approaching the subject, contact with the ore is very difficult to obtain. If wet pulp is introduced at the bottom of a bath of mercury, it rises to the surface in lumps, surrounded by films of water, and dry pulp is still more effectively protected from contact with the mercury by films of air. If a thin stream of pulp is run over the surface of a bath of mercury of sufficient size, the chances of the particles of gold coming in contact with the quicksilver (by settling through the stream) are greatly improved, but in this case the more convenient copper plate could be substituted for the bath. Moreover, it is well known that rich gold or silver amalgam catches gold more readily than pure mercury, and, whilst the surface of the plates can readily be covered with such pasty amalgam, it would involve a larger and unnecessary sinking of capital to keep any considerable percentage of precious metals in the baths. For these reasons, and for the practical reason that plates are found to work better than baths, the use of the latter has been gradually more and more restricted, until they are now only to be found in narrow wells and riffles for the purpose of catching hard amalgam and floured mercury, and as purely supplementary aids to the plates, whilst they are sometimes dispensed with altogether. When a proper disposition of the plates is made, it is rare to find amalgamable gold escaping into the tailings. Even if the latter contain several pennyweights of gold to the ton, this does not show that the amalgamation effected on the plates is unsatisfactory, and, where tailings have to be reground or roasted, or treated in any way that may alter the condition of the gold, before a further extraction by mercury is obtained, no proof is afforded that the plates are not doing their work.

CHAPTER VIII.

OTHER FORMS OF CRUSHING AND AMALGAMATING MACHINERY.

Special Forms of Stamps.—Since within certain limits and under certain conditions the capacity of a stamp battery depends on the number of blows given per minute and on the momentum of the fall, various contrivances have been suggested with a view to increase both of these.

In *pneumatic stamps,* such as the Husband and Phoenix stamps, a crank shaft raises the stem, which is attached to an air cylinder. The air in this is compressed, and the stamp head attached to a piston is thus raised. It is forced down to deliver the blow by similar means.

In *spring stamps,* such as the Elephant stamp, a crank shaft raises the stamps against the action of powerful springs, which increase the power of the blow. The rate of wear and cost of repairs have been found to be high in all these stamps, and they have never been much used.

Morison's high-speed stamp[*] is lifted like an ordinary pneumatic stamp, except that the compression cylinder contains water instead of air.

Steam Stamps.—The ordinary form of steam stamp consists of a direct-acting vertical engine, having a steam cylinder and slide valve at the top, the piston-rod being rigidly connected with the stamp. Each stamp head works in a separate rectangular cast-iron mortar, with screens both at the front and back, and sometimes all round. The screens are of Russia sheet-iron with punched holes of about $\frac{1}{8}$ to $\frac{1}{10}$ inch in diameter, the steam stamp being best adapted for coarse crushing. The speed of working is from 90 to 100 blows per minute, and the out-put is from 100 tons to as much as 225 tons of ore per head in twenty-four hours. It is obvious that gold could not be economically saved on plates inside the mortar of one of these stamps, and, as a matter of fact, until recently they were only employed in coarsely crushing the copper ores of the Lake Superior region. Nevertheless, the curious fact seems to be well established that these stamps, with their heavy blow, do not make so much slimes as the ordinary gravitation stamp. They have been tried in crushing silver ores in Montana, and gold ores at the Homestake Mine, in the Black Hills, through a 30-mesh screen. As their capacity is so great their use is limited to cases in which large quantities of ore are available, one Ball steam stamp, such as is used in the Lake Superior district, being equal to at least fifty head of gravitation stamps. They may be useful in preparing ores for re-grinding.

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[*] *Proc. N.E.C.I. of E. and S.*, 1896.

The chief advantages of the steam stamp are economy of space and labour. The advantage of subdividing the work among a number of batteries is that stoppages for repairs and breakages affect only a small part of the crushing capacity at one time.

The Huntington Mill.—The Huntington Roller Mill, here described as a type of the many good roller mills now in use, is best suited for the fine crushing of ores which are not too hard.

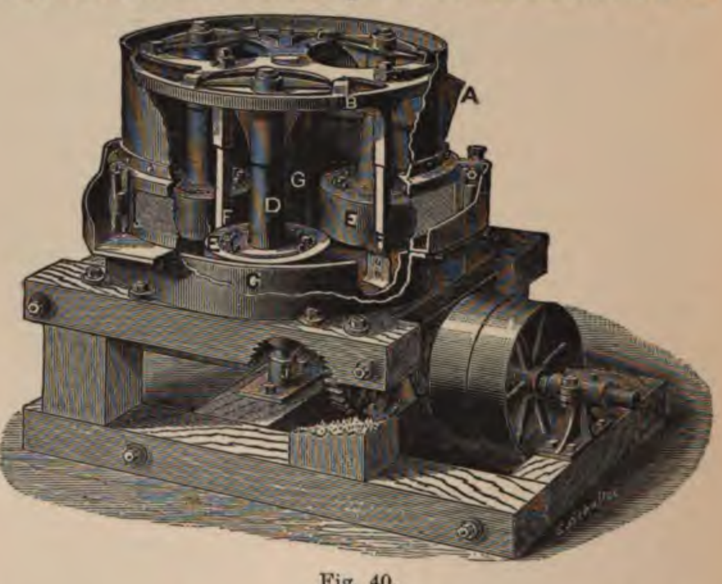

Fig. 40.

It consists of an iron pan, at the top of which a ring, B (Fig. 40), is set, and attached to this are three stems, D, each of which has a steel shoe, E, fastened to it. The stems are suspended from the ring and are free to swing in a radial direction, as well as to rotate round their own axes, whilst the whole ring, B, with the stems and shoes, revolves round the central shaft, G. The shoes or rollers, as they are called, are thus driven outwards by centrifugal force and press against the replaceable ring die, C. In front of each roller is a scraper, F, which keeps the ore from packing. The rollers are suspended with their bases at the distance of 1 inch from the bottom of the pan, which can also be replaced when worn. The lowest part of the screen is situated a little above the top of the rollers, and outside it there is a deep gutter into which the ore is discharged, and from which a passage leads to the amalgamated plates. The ore and water being fed into the mill through the hopper, A, generally by an automatic feeder, the rotating rollers and the scrapers throw the ore against the sides where it is crushed to any required degree of fineness by the centrifugal force of the rollers acting against the ring die. From 17 to 25 lbs. of mercury are placed in the bottom of the pan, the clearance below the rollers permitting them to pass freely over the mercury without coming in contact with it, so that it is not stirred up and "floured," but the motion is such as to bring the pulp in contact with the quicksilver. The speed of the mill is from 45 to 75 revolutions per minute. The ore should be broken in rock-breakers to a maximum size equal to that of a walnut, or, better still, of a cobnut, before being fed in. The action of the rollers is one of impact rather than of grinding, the ore being granulated without the production of much slimes. The free gold, as soon as it is liberated from its matrix, is in great part amalgamated and retained by the mercury at the bottom of the pan, the remainder being caught on the plates outside the mill. Coarse gold is caught inside and fine gold outside the mill, but the yield inside is comparatively small when ores with high percentages of sulphides are in course of treatment.

The mill is particularly adapted for the treatment of ores containing brittle sulphides, which, if pulverised by stamps, are liable to become "slimed," and so to be in an unsuitable condition for concentration. It is also suitable for argillaceous quartzes, which yield their gold more readily under the "puddling" action of the rollers than when pounded by stamps. Moreover, the Huntington mill does much more satisfactory work than stamps on soft ores or in regrinding coarse tailings. The reason for this lies, of course, in the relatively large amount of screen area in the mill and its consequent high efficiency of discharge, a point in which stamps are decidedly inferior to it. As the splash is heavy against the sides the wear of the screens is somewhat rapid, but they can be very quickly replaced.

The capacity of a 5-foot mill, the one which is most commonly in use, is from 10 to 20 tons of rock per day through a 30-mesh screen, the power required being from 10 to 12 H.P. The weight of each of the roller shells, which are replaceable, is about 170 lbs. The wear and tear on the replaceable parts is very great, amounting to about 14 ozs. per ton of rock crushed when soft ores, previously broken small, are being treated. If large pieces of hard quartz are fed into the mill, or if the mill is overfed, the mercury is splashed against the screens and passes through with the pulp, and when by accident pieces of iron or steel are introduced, the ring die is occasionally broken.

Another source of disaster in Huntington mills lies in the use of acidulated water, such as that derived from mines or encountered when decomposing pyritic ores are treated; the mill is rapidly corroded and rendered unfit for work by such water.

The chief advantages supposed to be gained by the use of Huntington mills instead of stamps may be thus epitomised:—

1. *Reduced First Cost.*—The cost for the same capacity is not more than two-thirds that of stamps, even at the manufacturers' shops, while the difference in favour of the mill is even more in outlying districts from its light weight, and corresponding low freight, and from the cheapness of its erection.

2. *Saving of Power.*—The mill is said to run with about one-half the power per ton of ore crushed.

3. *The wear and cost of renewals is less* for the mill than for stamps, the cost being from twopence to threepence per ton of ore for the former, against about fivepence or sixpence for the latter.

4. *There is less loss by flouring* of amalgam and quicksilver, while the good discharge and absence of grinding leaves the pulp in a better condition for concentration.

The first three advantages appear to refer only to such soft and brittle ores as are especially suited to the Huntington mill. The mill requires to be set to work in an intelligent manner by experienced and skilful hands, and watched carefully. The dangers of over-feeding have been already alluded to. One difficulty in automatic feeding is that self-feeding, such as is carried on by stamps, is impossible. The automatic feeder must work separately and be set to feed a certain weight of ore per hour, this weight having been determined by trial. If, after this, there is any change in the hardness of the rock, no automatic change in the rate of feeding takes place, and the machine may be choked up or run at below its maximum capacity unless watched and the feeder regulated. Another difficulty is in the quantity of water to be added. An excess of water, making thin pulp, does not favour internal amalgamation, and it may be stated that, in general, the pulp should be kept as thick as possible, consistent with its prompt discharge through the screens when sufficiently fine. If the pulp is too thick to run easily over the copper plates outside the mill, water may be added there by means of a perforated pipe. The rate of running should be as high as possible, since, if the other conditions are the same, the crushing power varies as the cube of the number of revolutions per minute. The usual rate is from 70 to 100 revolutions per minute.

A few examples are appended of the results obtained in actual practice by this excellent machine. At the Spanish Mine, Nevada County, California, there are four Huntington mills, three of 5 feet diameter and one of 4 feet diameter. The ore is free-milling and is passed through a Blake stone-breaker and thence to the mills. The four mills run at 58 revolutions per minute, and pulverise 35 tons of ore each in twenty-four hours, to pass through a slot screen equal to 20 mesh. The pulp is passed over the usual amalgamated plates after leaving the mills, and ¼ oz. of mercury is added with each ton of ore. Forty-five per cent. of the gold recovered comes from the inside of the mill, where the amalgam obtained is much richer than that from the plates. The loss of quicksilver is from $\frac{1}{15}$ to $\frac{1}{30}$ oz. per ton of ore, and the total cost of milling is about one shilling per ton, while for the month of November, 1887, it was only tenpence per ton. The ore is a soft talcose slate, containing streaks and veins of ferruginous quartz, carrying gold. The chief trouble in working lies in the frequent re-adjustment of feed which is found necessary. In a special test run of one month 42·4 per cent. of the gold contents was extracted, and the remainder lost in the tailings. This poor result was probably due to over-feeding, but profits were made, nevertheless, although the ore yielded only a little over 1 dwt. of gold per ton in 1887 and 1888. Twenty-two horse-power were used by the Huntington mills in crushing from 120 to 140 tons per day.

At the Shaw Mine, El Dorado County, a 5-foot mill, making 50 revolutions per minute, pulverised 10 to 12 tons per day, so as to pass through a 25- to 30-mesh screen. At the Mathines-Creek Mine, in the same county, a 5-foot mill pulverised 9 to 10 tons per twenty-four hours, so as to pass a screen equal to a 40 mesh. At the Monto Cristo Mine, Mono County, two 5-foot mills, running at from 65 to 75 revolutions per minute, pulverised 2 tons of ore per hour to pass a screen equal to a 40 mesh; 25 lbs. of quicksilver were charged into each mill at the commencement of the run, and about ½ oz. more added each half hour.

Although cases have been adduced in which 90 per cent. of the gold contents of an ore crushed in the Huntington mill was retained inside the machine, this is decidedly exceptional, and the mill is probably inferior to the stamp mill as an amalgamator on many ores, although its product is better adapted for treatment on copper plates and by concentration.

Other Roller Mills.—In the *Griffin Mill*, there is a single roller suspended in a similar way to the rollers in a Huntington mill. The roller is about 18 inches in diameter, and rolls on the inside of a die ring 30 inches in diameter. It takes material up to 1½ inches in diameter, and crushes either wet or dry. It does not appear to be used anywhere either as an amalgamator or to prepare ore for amalgamation, but is described here for convenience. In December, 1904,* 34 Griffin mills and 14 Huntington mills were in use in Western Australia preparing ore for cyaniding.

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* *Government Gazette of Western Australia*, December, 1904, pp. 530-554.

Among other roller mills, the *Bryan Mill* is one of the most successful.* It resembles the Chilian mill and consists of an annular mortar in which are fixed segmental annular steel dies. The ore is crushed by three vertical rollers with fixed horizontal axles, which rotate in journals fixed to an annular horizontal rotating plate. This plate revolves round the central axis of the machine, and the rollers are made to run round on the dies. A scraper follows each roller to keep the dies clean and to discharge the pulp through the screens which form the wall of the trough all round. The pulp runs round at the rate of 300 feet per minute or more at the periphery, the plate moving the rollers making 30 revolutions per minute. The mill is suitable for hard as well as for soft ores. The capacity of a 4-foot mill is about 15 or 20 tons of quartz ore per day through a 40-mesh screen. The Bryan Mill has been introduced on the Pacific Coast, in Mexico, and in Australia for crushing gold ores.

The *Chilian Mill* (see also p. 93), formerly used in the Patio process and for preparing ores for barrel amalgamation, has recently been re-introduced into the United States for crushing gold ores requiring very fine pulverisation. In one mill, sent to Mexico, the rollers were 8 feet in diameter and weighed 10 tons each, the shell or tire being 6 inches thick. These made 12 revolutions per minute, and the capacity was 18 tons of ore per day through a screen of 120 mesh. They are more usually employed for coarse crushing.

The Tyrolean Mill.—In Hungary, in Transylvania, and in the Tyrol, the principle of separation of the operations of crushing and amalgamation is still successfully used, the mills employed having a strong resemblance to the amalgamation pans described below (pp. 164-169). In some districts of Hungary and Transylvania, Californian stamps have been recently introduced, displacing to some extent the square German stamps, but the amalgamation is still effected in bowls or pans, which are the modern equivalents of the old Tyrolean mills still to be found at work in certain retired valleys in the Eastern Alps (see p. 98). The best known of these modern machines are the *Schemnitz Mill* and the *Lazzlo Amalgamator*.

The *Schemnitz Mill*, shown in Fig. 41, closely resembles the ancient form and is always used in pairs, one overflowing into the other. It consists of a cast-iron bowl, about 2 feet in diameter at the top and 18 inches at the bottom, with an internal depth of 7 inches. A massive wooden muller, B, hollowed out inside in the form of a cone, is suspended in the bowl by the iron rods, C, and is revolved at the rate of about twenty turns per minute. About 26 lbs. of mercury are charged into the bowl, and the lower face of the muller is set with twenty iron teeth, D, placed radially, which almost touch the mercury and force the pulp (delivered into the mill from the trough, E) outwards towards the periphery of the bowl, whence it overflows into the second mill.*

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* See also article by F. Furman in the *Metal Miner*, 1896, p. 266; and E. A. Tays, *Trans. Am. Inst. Mng. Eng.*, vol. 29 (1899), p. 776.

In the neighbourhood of Schemnitz, in Hungary, lead ores, containing 2 to 4 dwts. of gold per ton are crushed by stamps and are treated in these mills, the tailings being concentrated for smelting. At Nagybanya, in Hungary, the loss of mercury in these mills amounts to from $\frac{1}{2}$ to 1 part for each part of bullion recovered. Thus at the Kreuzberg works, near Nagybanya, in one month 155 metric tons of ore were treated, and 1·957 kilogrammes (62·9 ozs.) of gold were recovered with the loss of 1·9 kilogrammes of mercury, and in another month 754·5 metric tons of ore yielded 5·756 kilogrammes (185·8 ozs.) of gold, with the loss of 2·9 kilogrammes of mercury.† The clean-up usually takes place once or twice a month, and the capacity of each pair of mills is from $\frac{3}{4}$ to 1 ton of ore per day.

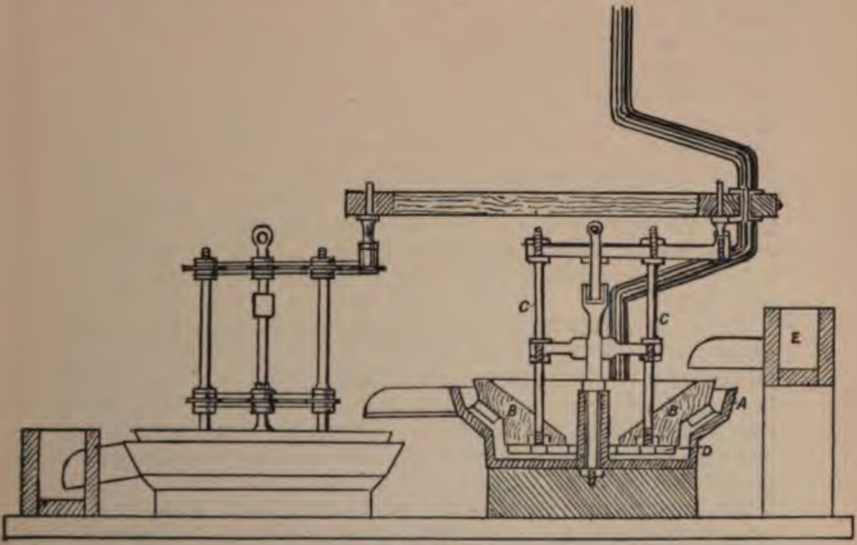

Fig. 41.

The *Lazzlo Amalgamator*‡ has only lately come into use, but has already proved itself to be of value in Transylvania. It differs from the Schemnitz mill mainly in having a flat-bottomed bowl, which is furnished with two circular iron partitions dividing the bowl into three concentric compartments. The ore is fed into the centre one and overflows into the others in succession, and thence into a second smaller bowl. The muller is of iron, and dips into each compartment of the bowl compelling the pulp to pass down and come in contact with the mercury three times before it escapes, and the gold, owing to its density, does not readily pass upwards and over the partitions.

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* *Traité de Metallurgie*, by C. Schnabel, translated by Gautier, p. 738.

† *Loc. cit.*

‡ *Ibid.*, p. 739.

At the Füzesd Dreifaltigkeit Mine, near Boicza, in Hungary, the ore is crushed by Californian stamps, the capacity of which is 0·8 ton per head in 24 hours, and then passed through Lazzlo amalgamators. The amalgam is collected in settling pans, which resemble the amalgamators, but have no iron teeth, and the tailings are classified in spitzkasten, and concentrated by passing over buddles and then over canvas tables. The auriferous pyrites is caught on the buddles and sold to smelters, but the product of the canvas tables is very rich in free gold, and is ground with mercury in iron mortars by hand with pestles. The Lazzlo amalgamators are about 25 inches in diameter, and each pair treats from 1·7 to 2 tons of ore in twenty-four hours, 75 to 80 per cent. of the gold being saved. The loss of mercury is about 1 oz. per ton of ore, and the power required for twenty-four pairs of amalgamators and eight settlers is 4 H.P.

At a mine near Brad, in Transylvania, each pair of amalgamators treats from 3 to 3½ tons of ore per day, but only 55 per cent. of the gold is extracted, the tailings being treated in American pans, and concentrated on Bilharz tables for smelting. Specimens of the ore containing visible gold are amalgamated by hand in mortars, and the rest contains about 8 dwts. per ton.

Amalgamation Pans.—An amalgamation pan consists of a circular cast-iron pan, provided on the inside with a renewable false bottom of cast-iron—constituting the lower grinding surface—and a "muller," or upper grinding surface (*d*, Fig. 42), attached to a vertical revolving spindle, *g*, which is set in motion by bevel wheels, *t*, placed below the pan. The muller grinds to impalpable pulp ore which has been already reduced to a coarse powder by stamps, and also mixes the ore with mercury, introduced into the bottom of the pan, and so amalgamates the gold and silver. The origin of the pan is probably to be traced to the Mexican arrastra, and some of the varieties of the pan are merely slightly modified arrastras. One variety consists of a sectional wrought-iron pan fitted with a granite grinding bottom and with granite mullers, which are attached to a vertical spindle rotated by hand or by animal power.

In work on gold ores the use of amalgamating pans was formerly mainly limited to regrinding skimmings, blanket sands, and concentrates obtained in working a stamp mill. Pans are now largely used at Kalgoorlie in grinding roasted sulpho-telluride ore (see Chap. xvi.). In this case some gold is amalgamated, but the main object of the pans is to prepare the ore for cyaniding. The subject is dealt with in Chapter xi.

Silver ores are often crushed in a battery, roasted with salt if necessary, and then amalgamated in pans. Silver ores containing considerable quantities of gold are often similarly treated, but with purely gold ores it is seldom necessary to resort to this process, and only a brief account is given below.*

Gold ores which do not yield a fair percentage of their values when run over amalgamated plates are occasionally treated in pans. In such cases the ore may be roasted or treated raw. As already stated (p. 148) it is seldom advantageous to roast a gold ore before amalgamation, since, although in a roasted pyritic ore specks of free gold may often be detected where none were visible in the raw ore, a part of the precious metal usually appears after roasting to be difficult to bring in contact with mercury. The cause of this is not always easy to discover, but it may sometimes be due to the coating of gold by thin films of iron oxide or other mineral. Moreover, the addition of salt to a gold ore in the roasting furnace, as is pointed out in the chapter on chlorination, is often attended by appreciable losses by volatilisation. These two causes are sufficient to account for the low percentage of gold usually extracted when an auriferous silver ore is treated by roasting with salt and pan-amalgamation. Under exceptional circumstances a gold ore may prove to be satisfactorily handled by roasting and amalgamation.

Pan-amalgamation, whether the ores treated are raw or roasted, may be conducted in one of two ways. The older system is to crush wet in the stamp mill, and collect the ore in large shallow settling pits or pointed boxes (see p. 172). A sufficiently dry pulp having been obtained by draining, it is dug out by hand and charged into the pans. A newer method, the continuous system, is briefly described on p. 168.

Old System. — The amalgamating pans in use are very numerous, and vary greatly in form. The shape of the bottom was formerly much in dispute, flat, cone-shaped, and hemispherical bottoms each having its advocates, but it is now generally believed that flat-bottomed pans are the best, wearing more evenly and doing more work. The pans are often heated, so as to increase the rate of amalgamation, by means of steam led through a chamber below a false bottom in the pan, but the more economical device of introducing steam into the pulp itself has also at all times been in use. The objections to the latter course are that the pulp may be so much diluted that amalgamation is checked, and that oil is liable to be introduced with the steam with equally disastrous results. When the ore is roasted before being treated in the pan, it is in some mills charged in hot, hot water being added also, and as the pan is covered up and is still warm from the previous charge, it remains at a sufficiently high temperature throughout the operation without further treatment. The grinding of the ore by the muller is an additional source of heat.

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* See H. F. Collins' *Metallurgy of Silver*, 1900, pp. 80-111, for a detailed description of the Pan-amalgamation Process.

One of the common forms of amalgamating pans is shown in Fig. 42. This pan is 5 feet in diameter, with cast-iron bottom, *a*, and wooden sides, *h*. The mullers are shown resting on the cast-iron dies, *c*, which protect the bottom from wear, whilst replaceable shoes attached to the lower surface of the mullers are also shown. The shoes and dies can be kept in contact while the spindle, *g*, is rotated, so that the ore can be ground, or the muller can be raised by rotating the hand-wheel and centre screw, *j*, on the top of the spindle, so that only circulation

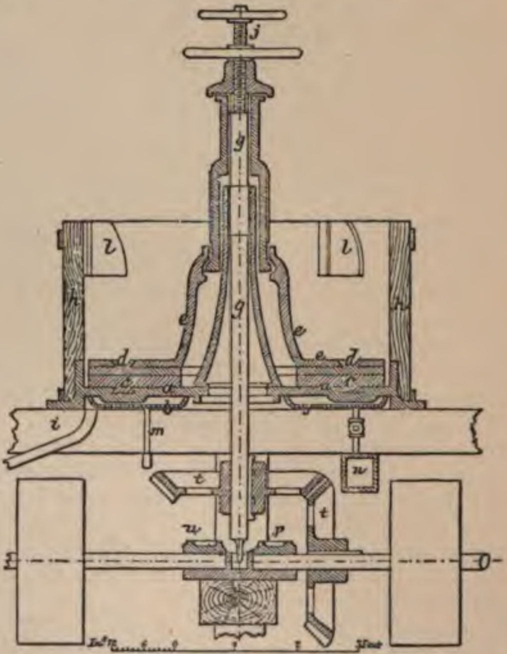

Fig. 42.

and mixing of the charge take place. In some pans copper plates, *l*, are introduced, being attached to the side walls and projecting into the interior. These plates are intended both to mix the pulp and to catch the amalgam, much of which is retained on them. The more usual system is to employ separate vessels called settlers for the collection of the quicksilver and amalgam, after the pans are discharged. The speed of the muller is usually from 65 to 75 revolutions per minute; below the muller the pulp is continually worked from the centre of the pan to the circumference, being returned towards the centre above the muller and passing down through the latter by inclined slots which terminate near the centre. In Fig. 42, which represents the form known as the Patton pan, *n* is the main through which steam is passed into the chamber, *b*, to heat the pulp, and *m* is the outlet pipe.

The method of operation is as follows:—The charge of ore is introduced with the mullers raised slightly and kept revolving, water being added at the same time in quantities sufficient to make the pulp of a pasty consistency, so that globules of mercury remain suspended in it without subsiding. The mullers are then lowered and the ore ground for from two to four hours, after which the mullers are raised and the mercury added gradually, and thoroughly mixed with the pulp for six to eight hours longer. The object in raising the mullers is to prevent the sulphides from being ground up with mercury, which would cause considerable losses by flouring and sickening. Nevertheless this raising of the mullers is not an invariable practice. When the amalgamation is thought to be complete, water is introduced to dilute the pulp, and the whole is discharged into a settler; or else the diluted pulp is stirred by the raised muller at a reduced rate of speed until the globules of mercury have re-united and sunk to the bottom, when the pulp is gradually run off, beginning at the top, usually by pulling out in succession plugs set in the side of the pan at different levels. The discharge takes place into a bucket or tub, where some of the mercury accidentally carried over is caught. The bulk of the mercury in some mills is drawn off from the bottom of the pan before the pulp is discharged.

In order to facilitate amalgamation various chemicals have been recommended as desirable additions to the charge. At the present day it is recognised that most of these are either useless or absolutely harmful, and only salt, sulphate of copper, nitre, cyanide of potassium, lime, and sodium amalgam are now used. In treating gold ores, cyanide of potassium and sodium amalgam are added to keep the mercury clean and lively, but the latter chemical is now comparatively rarely resorted to. Salt and sulphate of copper are chiefly added to silver ores, their use having been suggested by the Patio process. They are believed to decompose certain base minerals, and so to prevent the sickening of mercury, which would otherwise be caused by their presence, and also to liberate silver from some of its compounds and thus render it capable of amalgamation. The use of lime is of course to neutralise any acid sulphates of iron, &c., which may be formed by the partial decomposition of the ore, and so to prevent the sickening of the mercury. If added when the pulp is diluted, lime is said to be efficacious in assisting the mercury to collect together and settle.*

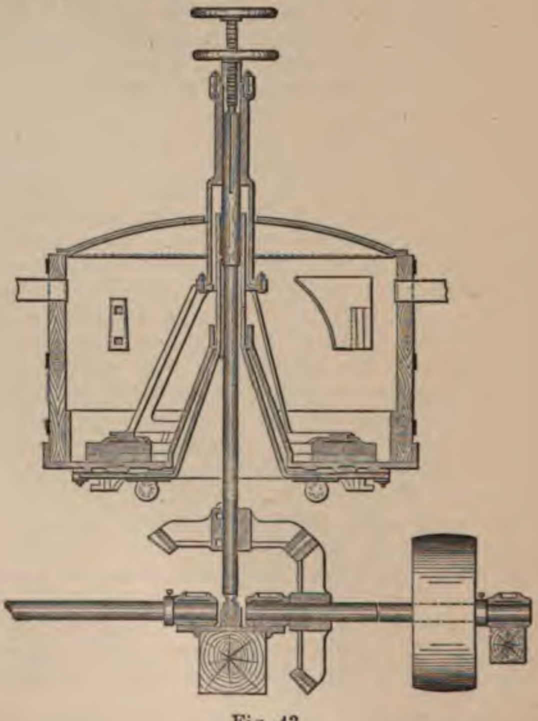

Fig. 43.

Scale, 1 inch = 3 feet.

The Boss Continuous System.—In this system of pan-amalgamation the pulp is continuously run direct from the stamp battery through a series of pans arranged so that each overflows into the next one, which is placed at a slightly lower level. The first two or three pans are arranged as grinders, the battery pulp not being fine enough for complete amalgamation, and the pulp is then passed through a series of amalgamating-pans supplied with mercury, after which the mercury and amalgam are separated from the ore in settlers, which are larger pans in which the pulp is diluted and stirred less vigorously. The tailings overflow from the settlers, and are run to waste or led over concentrators. The number of pans arranged in series through which the pulp must pass, in order to yield a fair percentage of its precious metals, is determined by experiment for each particular ore. It is obvious that the consistency of the pulp must be thinner than has usually been considered desirable for successful amalgamation, and, as a matter of fact, its volume is usually doubled by the introduction of the continuous process, but in spite of this the percentage of extraction is not lower than by the old method. By the Boss system there is a large saving in labour, fuel, and in wear and tear; the settling pits or pointed boxes are dispensed with, and no movement of the pulp by hand is needed. The mercury is collected in wells and pumped up into tanks, whence it is fed automatically into the amalgamating pans. One of the pans in use in this process is the Boss Standard Pan, shown in Fig. 43. It will be noticed that there is a steam chamber below the false bottom of the pan, extending up into the conical space in the centre, for warming the pulp.

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* For a full account of the chemical reactions involved in pan-amalgamation, the student is referred to the *Report of the United States Survey of the Fortieth Parallel*, vol. iii., chap. v.

Treatment of Concentrates in the Pan.—The treatment of concentrates by pan-amalgamation is a survival of old methods, and does not represent the most modern practice, in which concentrates are either smelted or treated by wet methods. Whether they have been previously roasted or not, the treatment of concentrates in pans is seldom attended by the successful extraction of a high percentage of the gold. A stone arrastra usually gives better results in treating roasted concentrates than an iron pan. In Australia the method is often adopted of employing a large excess of mercury and little water, and of keeping the roasted material from contact with iron, and in some experiments conducted in Mexico, C. A. Stetefeldt found that by the use of gold amalgam instead of mercury, and by grinding in stone vessels, a high percentage of gold was extracted from low grade ores.

Among special forms of pans designed to treat concentrates are the Berdan pan and the Britten pan, which were both introduced about fifty years ago.

The *Berdan pan* is a shallow annular basin about 4 feet in diameter, surrounding a cone which is attached to a spindle set at an angle of about 15° to the vertical. The spindle is rotated by bevel gearing at 20 to 30 revolutions per minute, and carries the basin round with it. In the annular basin are one or two loose iron balls which remain at the lower side of the cone when the pan revolves. The pulp is fed in with mercury at the higher side, and is ground by the balls and discharged over the lower edge of the pan. The capacity of the Berdan pan is from 1 to 2½ tons per day. It is used chiefly in Australia.

The *Britten pan* is a deeper cast-iron stationary basin almost hemispherical in shape, in which a pear-shaped muller rolls round. It is used for grinding and amalgamating rich specimen ore and concentrates in Wales. It is of small capacity, and is not a continuous machine like the Berdan pan, but grinds the charge put into it as long as is desired.

CHAPTER IX.

CONCENTRATION IN GOLD MILLS.

Concentration.—The object of concentration is the separation of the heavy valuable mineral from the light worthless gangue. Complications are often introduced by the fact that various base minerals must be separated from one another, an ore being subdivided into several products. Most gold ores, however, only require separation into two parts—the "concentrates," in which the precious metal is contained, and the "tailings," which are thrown away. The German system of coarse crushing, sizing by means of screens, and concentrating on jigs, is not, as a rule, applicable to gold ores proper, although much used on auriferous lead, zinc, and copper ores. This system will not be described here, where only the methods in use for the treatment of battery sands, after passing over the amalgamated plates, will be considered.

All the concentrating machines depend for their action on the effect of a difference of densities on the fall of bodies in a fluid. The fluid employed in almost every instance is water, although several machines have been devised in which air is used as the concentrating medium. It has often been proposed to use some solution which shall have a lower density than the valuable mineral to be saved, but a higher density than the worthless gangue; the mineral would then sink, while the gangue would remain floating. The high cost of any such solution is sufficient to put this method out of the question, without discussing any further disadvantages.

The fall in water of solid materials takes place according to two laws, one applicable to very shallow water, through which the particles fall with increasing velocity, while the other is true when the depth of the water is considerable, so that the particles for the greater part of their course proceed at their maximum velocity. In shallow water the fall is almost entirely according to density, so that those machines which utilise only the first instants of the fall will have great efficacy in concentrating. It is this fact which has necessitated the use of shallow currents in concentrating tables, sluices, &c. In almost all these machines the fine sand and slime is brought into suspension in water, and the liquid is then run over an inclined surface. The deposit of sand, which is thus formed on the table, tends to become enriched in heavy minerals, because the stream moves faster at the surface of the water, where the lighter particles remain, than it does next the bed, where the heavy particles have settled. The deposit is continually worked up and brought again into suspension by a rake or broom, or by a series of shakes or blows imparted to the apparatus, so that the effect mentioned above is repeated frequently. If the stirring up is violently performed all the slime and very fine particles are kept in suspension in the water and carried away and lost, a slow stream of water and very slight agitation being favourable to their retention in the deposit which is formed. When the stream of water is rapid and voluminous, fine material, whether heavy or light, is swept away and lost in the tailings, whilst if too small a stream of water is used, much worthless sand is deposited with the concentrates. It follows that the amount of water used must be regulated according to the work which it is proposed to do. Frequently, it happens that clear water must be added to the pulp to dilute it sufficiently. On the other hand, it occasionally happens that the pulp is too thin, especially after hydraulic sizing, and water is then removed by means of *settling boxes*, described below.

Another operation, which it is often of the utmost importance to perform before concentration, is that of classification by size. The necessity of this is obvious, when it is remembered that a shallow stream of water swift enough to carry down fine sulphides, might be powerless to move a pebble of quartz. The usual method of classification is based on the varying rates of fall of particles through a deep column of water. In this way *equal-falling* particles are obtained together, and since a sphere of galena is equal-falling with a sphere of quartz of from 1·5 to nearly 4 times the diameter, according to the absolute sizes,* it follows that the sizing cannot be perfectly performed by this method. Nevertheless, such sizing as is possible in this way, when efficiently performed, is of great assistance as a preparation for treatment by shallow-stream concentrators. Sizing by screens is more perfect, and may come into more extended use.

Settling Boxes.—These are of the same shape as the spitzkasten or pointed boxes introduced by von Rittinger. They are for the purpose of allowing the sand and mineral in suspension in a flowing current to settle, so that the part of the water not needed in the subsequent treatment of the ore may be run off; if it is desirable, this excess of water is available for use over again. These boxes will deliver a small stream of thick concentrated pulp at the bottom and give an overflow of almost clear water at the top. The boxes manufactured by Messrs. Fraser & Chalmers are shown in plan and elevation in Fig. 44. The main points observed in the construction of these boxes are as follows:—

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* See *Ore Dressing*, by R. H. Richards, 1st edition, p. 471. Richards states (*Mineral Industry for 1896*, p. 705) that in practice, with particles between 10- and 60-mesh screens, the diameter of the quartz is from 2¼ to 3¾ times that of the galena, varying with the absolute size.

The sides are at an angle of at least 50° to the horizontal, to insure the uninterrupted descent of the slimes. The pulp is delivered evenly across the end of the box by means of some such arrangement as the movable tongues (*a*, Fig. 44). Surface currents are prevented and the incoming pulp thoroughly mixed with the mass of the water in the box by the partition, *b*, which extends across the box near the inflow end. The discharge is made by cutting the other end of the box from 2 to 3 inches lower than the sides; this overflow is made perfectly level so that the water flows out in an even sheet. The discharge pipe for the pulp is at the bottom of the box. The siphon discharge shown is the most convenient, as the aperture need not be so much contracted as if direct discharge is used; and, besides this, less fall in the mill site is required. A contracted orifice is liable to be choked up, and a smaller diameter than 1½ to 2 inches is to be deprecated.

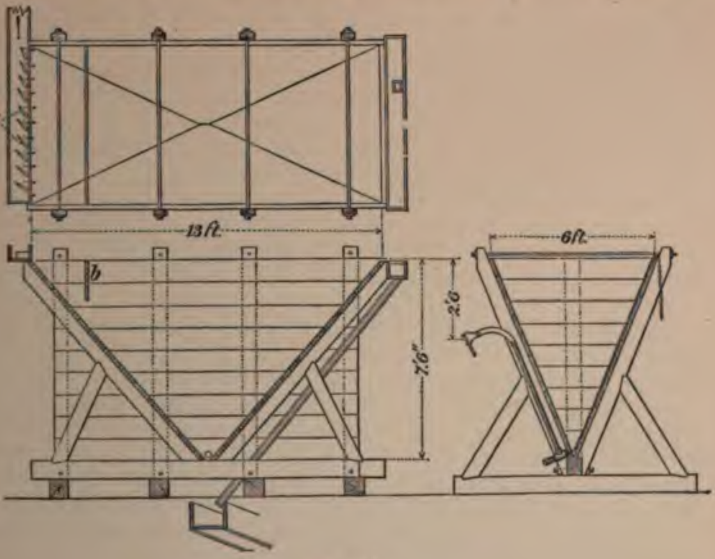

Fig. 44.

These boxes have the advantage over the ordinary settling-pits used to retain tailings and to catch pulp for pan-amalgamation, that they do not require to be dug out, while the settling, owing to the elimination of surface currents, is more perfect. One large box of 12 feet by 6 feet at the top and 7 feet 6 inches deep is sufficient to settle the product from five or ten stamps, according to the degree of separation of water and ore required.

If the pulp is required for treatment in amalgamating pans, such settling boxes as ordinarily used do not give material of sufficiently thick consistency for the purpose. To adapt them to this, M'Dermott suggests* "that the pulp from the battery should be spread out and flow across a long narrow box with sloping bottom, up which the settled heavier sand is moved by scrapers attached to a revolving chain. The sand is thus scraped out of the box in a thick condition, with very little accompanying water, and joins the thickened slimes from a large spitzkasten which receives the overflow of the narrow box above it. The spitzkasten alone has been used successfully in Mexico for thickening the whole pulp sufficiently for pan-amalgamation, by the use of the Ayton intermittent discharge opening at the bottom, which is reported to deliver a pulp of less than 50 per cent. water contents. The intermittent opening allows of a comparatively large discharge orifice for the required rate of delivery, thus lessening the danger of blocking."

Sizing by Screens.—Screening to equal sized particles is a desirable preparation for concentration on most machines, but in the past has not usually been employed on material finer than about 8 mesh, owing to difficulty in working, and to wear and tear of fine screens if used wet in the usual way. The two ordinary types of screens are the cylindrical revolving and flat shaking forms. For the finer sizes it has been usual to substitute hydraulic classifiers—*i.e.*, boxes with ascending currents of water, described below—for screens; but the one is not an exact equivalent of the other. Screens produce classes of particles of nearly equal size regardless of their respective specific gravities. Classifiers produce classes of equal falling particles, the sizes of which depend on their respective specific gravities. As a consequence the preparation for most concentrators is not so good by classifiers as by screens.

Lately several new devices have been brought forward for using much finer screens than were formerly considered practicable. Steeply inclined and rapidly jarring flat screens, with sprays of water, are reported to give good results; and very fine slightly inclined screens submerged in water (with jarring motion, and a special construction for delivering the coarser oversize into clear water) have been recommended. The advantage of this last type is that no extra wash water is used, screening is more perfect under water than with washing sprays, and the wear of screens is reduced to a minimum. Such screens have been tried as fine as 60 mesh—that is, with 3,600 holes to the square inch.

Hydraulic Classifiers were introduced by Prof. P. von

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* Private communication to the author.

Rittinger in the middle of the last century for use in the Hartz, and from their shape were known as *spitzkasten* or *pointed boxes.* These boxes have the shape of inverted pyramids, the stream of unsized pulp entering at one side and flowing out at the other, whilst there is also a small discharge at the apex. The current slackening on entering the box, the heavier and larger particles in suspension at once begin to settle, and, escaping the influence of the current, fall quietly to the bottom of the box where they are discharged. In each successive box a number of approximately equal-falling particles are removed, the closeness with which the subdivision is made varying with the size of the box, and the corresponding extent to which the current carrying the pulp is checked on entering it. The larger the box the more material is collected by it, and, therefore, the more heterogeneous the particles caught. In a small box the material collected consists more nearly of truly equal-falling particles.

These equal-falling particles, however, are in any box, whatever its size may be, carried away through the aperture in the apex by muddy water containing material of all sizes down to the finest slime, and it was to eliminate this material that the *ascending current* was introduced. This is a current of clear water, which enters at the apex of the box (*spitzlutte* *) in greater quantity than can be discharged by the outflow near the same spot, so that there is an upward current of water into the box. The result is that no muddy water is discharged below, but only the particles of ore which have weight enough to drop through the ascending current, so that by regulating the strength of this, any desired class of ore can be obtained. In Fig. 45, a form is shown in which the sliding partition assists the settling, by causing the pulp to pass downwards, rapid surface currents across the box to the overflow being thus prevented. The discharge of the heavy particles is effected through A, the clear water pipe itself, by the arrangement shown. The launder above the box supplies the clear water current, and shows the head of water used, which must be kept constant to ensure uniformity of results.

The chief defect of the early forms of pyramidal boxes was that, as the area of the vat became larger and larger towards the top, the velocity of the ascending water naturally became less and less, so that many particles were able to settle down below the level of the overflow, but were stopped by the increasing force of the current, so that an accumulation of ore took place half way up the box, and ultimately became so great as to interfere with the classification. Several remedies have been devised for this defect, of which one of the simplest and most effectual is to make the box pyramidal below, but with vertical sides in the upper part. This construction is partly carried out in the box described above. A slime-pit is added to catch the stuff which is too light to settle in the boxes, in all cases in which these slimes are of sufficient value to pay for treatment. The number of boxes used depends on the number of classes of material which it is deemed advisable to make. Usually two or three classes are sufficient.

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* There seems to be some difficulty in the definition of spitzlutten. Richards (*Ore Dressing*, p. 421) states that the word signifies a pointed tube, and that Rittinger used it in this sense. "Lutte," however, appears to mean a "gutter in which gold is washed." It is certainly convenient to limit the use of the word spitzlutten to pointed boxes with an ascending current of clear water, and spitzkasten to boxes without such a current.

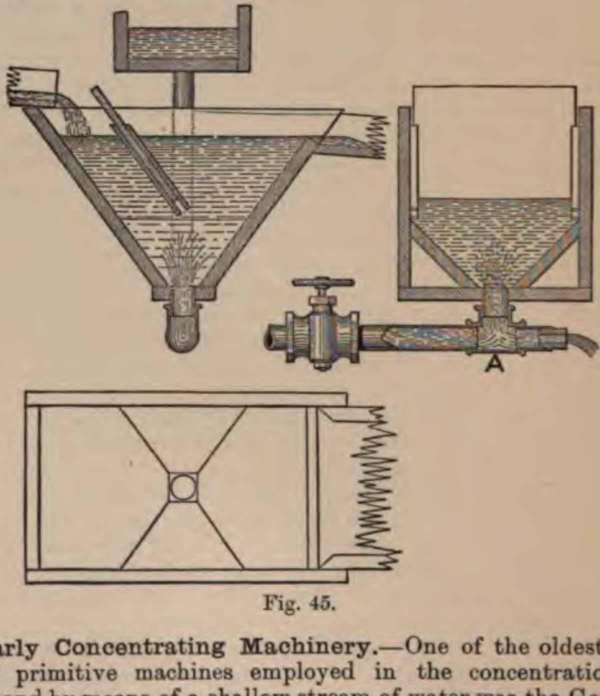

Fig. 45.

Early Concentrating Machinery.—One of the oldest and most primitive machines employed in the concentration of fine sand by means of a shallow stream of water was the German *buddle*, which has a distinct but imperfect resemblance to the Long-Tom described on p. 49.* Canvas tables were used below these buddles in Germany, and probably suggested the use of *blanket-strakes* or *tables*, which were adopted in the early days of the goldfields of the United States and Australia, and are still retained in some places. The rough surface of the blanketing seems to be particularly efficacious in catching and holding thin plates and spangles of free gold or of sulphides, which are readily washed off smooth surfaces by a current of water, and these rough appliances, although almost useless for catching slimes, still find favour where considerations of economy prevent the purchase of modern high-priced concentrators. The blanketing is usually in strips of 16 or 18 inches wide, and several feet long, and is nailed or stretched on wooden frames, which have an inclination of about ½ inch per foot.

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* For a description of the similar buddle formerly used in Colorado for the treatment of battery sands, see Raymond's *Mines, Mills, and Furnaces of the Pacific States.* New York, 1871, p. 357.

At intervals of about half an hour, when a quantity of mineral has already been collected on the rough surface, the blankets are taken off and washed in tubs of water, where a deposit collects which is afterwards dug out. At the St. John del Rey Mine, the framework supporting the blankets was hung on pivots above a shallow tank. When it was necessary to clean them, the framework was turned so that the upper surface of the blanket was inclined downwards, and the mineral washed off its surface by a hose, much time and labour being thus saved.

At this mine the trays supporting the blankets were 18 inches wide and 30 feet long, with a fall of 1 inch per foot. The upper 16 feet were covered with bullocks' skins, tanned with the hair on them, and in lengths of 26 inches; below these were a series of blankets or baize cloths of the same length, made of coarse wool with a long nap. The fall from the battery box upon the tray was 4 inches, a screen being placed across the end to break the fall of the water, and cause it to strike the tray nearly at right angles. About 90 per cent. of the gold contained in the ore was caught on these blankets. The blanket sands contained 95 per cent. of sulphides, and were so fine that 90 per cent. of them passed through a 100-mesh sieve. They were amalgamated in revolving wooden barrels, yielding 96 per cent. of their assay value, but this was due to the fact that very little gold was contained in the pyrites, most of it being present in the form of free particles.

Blanket sluices have been declared unsuitable for catching fine sulphides, and their concentrates are usually contaminated by admixture with much sand. If set at a proper inclination they will save fine amalgam and free gold, but even in this respect they are less satisfactory than shaking copper plates and riffles.

Riffled sluices were employed at the same time as blankets for effecting rough concentration. The riffles were formed of half-inch strips of wood nailed across the sluice box, the grade of which was about three-quarters of an inch to the foot. As soon as the concentrates had accumulated until they reached the top of the riffle, another strip was nailed on, and the process was repeated until the bed of concentrates was several inches thick, when they were scraped out and a fresh start made.

Similar to this device was the *raising-gate concentrator*, which was practically a riffled sluice in which the riffle was raised continuously by machinery, instead of being adjusted at intervals by hand.

The Round Buddle was invented in Cornwall, where it is still used in dressing the tin ores to the exclusion of almost every other concentrator. There are two varieties.

1. The convex round buddle, in which the ore and water are added at the centre of the machine, and flow down over the surface to the periphery.

2. The concave buddle, in which the pulp is added at the periphery of the machine and flows down to the centre.

In both cases revolving arms, carrying brushes, pass over the surface and stir the deposit as it is being formed, and the spouts disturbing the ore also rotate so as to deliver the pulp evenly. The buddles are from 12 to 18 feet in diameter.

These machines are not continuous in action, and after the deposit has accumulated to a depth of a few inches the operation is suspended and the deposit dug out. The "headings" or material on the upper 12 to 18 inches of the inclined surface are kept separate, and the stuff near the bottom of the slope is called the "tailings." Round buddles are not adapted to obtain a finished product in one operation. The headings and tailings must, as a rule, be subjected to further treatment, and between them is a large quantity of material which differs little from the original ore. Thus handling and re-handling of the stuff is necessitated, and it is on this account that these machines are not now much used on gold ores. The principle on which they depend is favourable to the collection of slimes, and the modern improved buddles are perhaps better adapted for fine ores than any other machines, except those employing a travelling belt. Modifications have been proposed to adapt these buddles to the treatment of gold ores by adding riffles containing mercury, and by other devices, but have not found much favour. One of the chief changes, which was proposed in the United States, is to keep the brushes and ore-spouts stationary, and to rotate the inclined bed.

Among concentrators with circular revolving beds, which were apparently suggested by the revolving buddle, may be mentioned Hendy's and Duncan's machines. These were formerly employed in California, but have given place to the vanners which are better adapted for saving slimes.

Percussion Tables.—In these machines the work of keeping the pulp in a state of agitation, done by the rakes or brushes in the German and Cornish buddles described above, is effected by sudden blows or bumps imparted sideways or endways to the table. The table is made of wood or sheet metal, the surface being either smooth or riffled.

End-bump tables are hung by chains or in some similar manner, so as to be capable of limited movement, and receive a number of blows delivered on the upper end. These blows are given by cams acting through rods, or else the table is pushed forward against the action of strong springs by cams on a revolving shaft, and then being suddenly released is thrown back violently by the springs against a fixed horizontal beam. The movement of the pulp depends on the inertia of the particles, which are thrown backward up the inclined table by the blow given to the table, the amount of movement varying with their mass, and depending, therefore, both on their size and density. The vibrations produced by the percussion also perform the work of the rakes in destroying the cohesion between the particles, and a stream of water washes them down. The result is that the larger and heavier particles may be made to travel up the table in the direction in which they are thrown by the blow, by regulating the quantity of water, while the smaller and lighter particles are carried down. These machines yield only two classes of material, headings and tailings. One such machine, the *Gilpin County "Gilt Edge Concentrator,"* was devised in Colorado, and has displaced the blanket sluices at almost all the mills at Blackhawk. It consists (Fig. 46) essentially of a cast-iron or copper table, 7 feet long and 3 feet wide, divided into two equal sections by a 4-inch square bumping-beam. The table has raised edges, and its inclination is about 4 inches in 5½ feet at its lower end, the remaining 1½ feet at the head having a somewhat steeper grade. The table is hung by iron rods to an iron frame, the length of the rods being altered by screw threads, so as to regulate the inclination to the required amount. A shaft with double cams, A, making 65 revolutions per minute, enables 130 blows per minute to be given to the table in the following manner; on

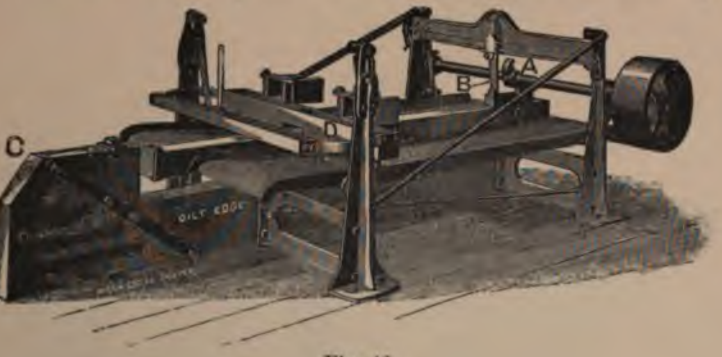

Fig. 46.

being released by the cam, the table is forced forward by the strong spring, B, so that its head strikes against the solid beam,

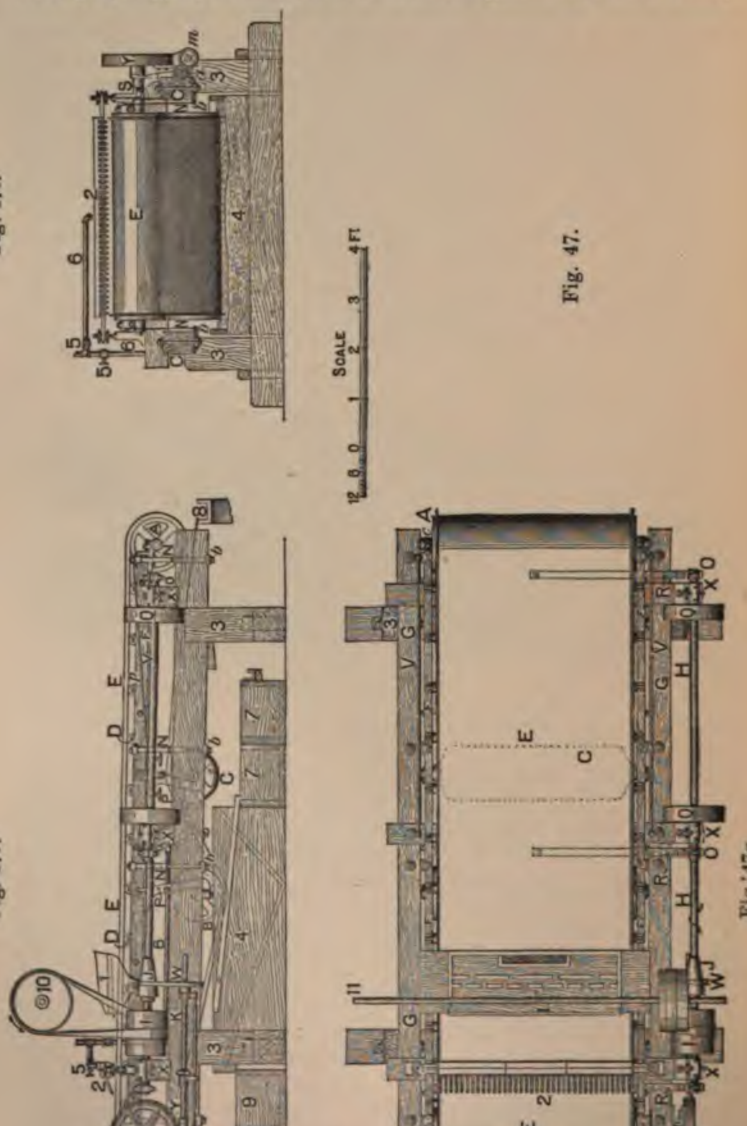
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Fig. 47c.

Fig. 47b.

Fig. 47.

Fig. 47a.

C, which is firmly united to the rest of the frame. The pulp coming from the copper plates is fed on to the table near its upper end by a distributing box, D, and is spread out and kept in agitation by the rapid blows. The sulphides settle to the bottom of the pulp, and are thrown forward by the shock, and eventually discharged over the head of the table at the left hand of the figure, while the gangue is carried down by the water and discharged at the other end. One machine is enough to concentrate the pulp from five stamps. If the table consists of amalgamated copper plates, it is of some use for catching free gold also, treating about 8 cwts. of ore per hour. This machine is not so effective in saving slimed pyrites as the Wilfley table or the vanners.

The Frue Vanner.—This machine is described in detail as being typical of the shaking travelling-belt concentrators. Machines of this class are especially adapted for treating finely-crushed battery sands which do not contain a large percentage of "mineral" (that is, sulphides and other heavy materials). They are frequently set to concentrate unsized pulp coming straight from the amalgamating tables.

The Frue vanner (Fig. 47) consists of an endless rubber belt, mounted on a frame, with its upper surface slightly inclined to the horizon, and subjected to two movements, a slow constant longitudinal movement and a slight and rapid side shake. The belt forms the bed or plane on which the dressing of the ore is effected, being an inclined plane, 12 feet long, and bounded down the two sides by projecting rubber flanges, which prevent the water and sand from dropping over the sides. An arrangement of rollers permits of the belt being slowly revolved in the direction of its length and *up* the incline; thus, though the dimensions of the working plane remain always the same, its surface is constantly travelling. The crushed rock in a stream of water is delivered near the upper end of the belt by means of the sand distributor, No. 1, Fig. 47, and flows down the belt towards its lower end. Now, as the inclination at which the belt is set is only from 3 to 6 inches on the 12 feet, and as the stream of water is not large and spreads over the whole width of 4 feet, it is obvious that, if it were not for the movements of the belt, much of the crushed rock contained in the water would settle on the belt, while the water and the finer and lighter particles of sand would alone reach the foot of the table and drop over into a waste launder.

In order to separate the heavy metallic minerals from the accompanying gangue or rock, a second stream of water is applied, whilst a gentle side shake is given to the belt, to keep the sand in a state of agitation, prevent it from "packing," and facilitate the sorting process. The water distributor, 2, is placed about a foot above the pulp distributor, and delivers small jets of water, 3 inches apart, over the entire width of the belt. The side shake thoroughly mixes this water with the pulp, spreads the whole uniformly on the belt, and enables the heavy particles of mineral to settle through the sand and cling to the belt, when they are carried up by it past the small jets of water and deposited in the collecting tank, while the lighter gangue is carried down by the stream and delivered into the tailings launder.

The Frue vanner is shown in plan in Fig. 47*a*, in side elevation in Fig. 47*b*, and in end elevation in Fig. 47*c*. The following description is abridged from that given by the manufacturers:—

A A are the main rollers that carry the belt and form the ends of the table, B and C being other rollers. The belt E, which is 4 feet wide and 27½ feet in entire length, passes through water underneath B, depositing its concentrations in the box, No. 4; and then passes over C, the tightening roller.

The cranks attached to the crank shaft, H, are ½ inch out of centre, thus giving a throw of 1 inch, which is the amount of the lateral throw. I is the driving pulley that forms with its belt the entire connection with the power. J is a cone pulley on the crank shaft, H. By shifting the small leather belt connecting J and W, the uphill travel of the main belt, E, is increased or diminished at will; the pulley, W, is moved by the hand-screw, *m*. R, R, R are three flat-steel spring connections bolted underneath the cross-pieces of the frame, F, and attached to the cranks of the shaft, H. These springs give the quick lateral motion—about 200 per minute.

No. 2 is the clear water distributor, and is a wooden trough which is supplied with water by a pipe, and the water discharges on the belt in drops through grooves 3 inches apart.

No. 1 is the ore-spreader, which moves with F, and delivers the ore and water evenly on the belt.

There is also a copper well that fits in (and shakes with) the ore-spreader as shown in the drawing. This is used in concentrating gold ores, for saving amalgam and quicksilver escaping from the silvered plates above, and can be taken out and emptied at any time. Into this well falls all the pulp from the battery. Its ends are lower than the wooden blocks of the spreader, so that the pulp passes over the ends of the well and is evenly distributed.

For some gold ores it is desirable to use on the ore-spreader a silvered copper plate the size of the spreader, and, when this is used, the wooden blocks of the spreader are fastened to a movable frame on top, so that they can be removed when the plate is cleaned-up, once or twice a month. No. 8 is a section of the launder to carry off the tailings.

The speed of the uphill travel of the belt varies from 2 to 12 feet per minute. If the ore treated be poor in pyrites, the upward motion of the belt does not exceed 20 inches per minute; if richer, the speed is increased accordingly, and in agreement with the inclination of the belt, being greater as this inclination increases, but usually not exceeding 3½ feet per minute. The inclination can be changed at will by wedges at the foot of the machine. The motion, the water used, the grade, and the up-hill travel is regulated for every ore individually, and must be adjusted with every change in the pulp, if good work is to be maintained.

The amount of water used on the machine is from 1 to 1½ gallons per minute of clear water at the head, and from 1½ to 3 gallons per minute with the pulp.

The capacity per day of twenty-four hours is usually put down as about 6 tons of material, fine enough to pass a 40-mesh screen. In California, in general, two Frue vanners treat the product of each battery of five stamps. Where the gangue is light, only one Frue vanner is sometimes used for five stamps.

Sizing of the pulp is usually omitted if the material is fine enough to pass a 30- or 40-mesh screen. If particles coarser than this are present they are removed by classifiers.

A riffled belt is sometimes used on the Frue vanner in cases where large quantities of comparatively coarse sulphides are present.

In the *Lührig vanner* the endless travelling india-rubber band is not flanged at the sides, and has a slight side inclination, which is, therefore, at right angles to the direction of travel; the latter is horizontal, not inclined; by an arrangement of cams and springs, end-blows are given to the framework carrying the belt. The pulp from the batteries is collected in settling boxes placed overhead and delivered on to the belt through a small distributing box situated above its right-hand upper corner, the belt being driven from right to left. Clear water is supplied through a perforated pipe fixed diagonally across the belt. The pulp moves across the belt from the higher side to the lower, this motion being assisted by the clear water; the light particles of gangue are washed down in this direction at a faster rate than the heavy particles of pyrites. On the other hand, the travel of the belt and the end-blows move the ore in the direction of the length of the belt. The results of the combined motions are as follows:—

1. Tailings pass off the table nearly opposite the distributing box at the right-hand end.
2. A middle product, containing both sulphides and gangue, is delivered near the middle of the side of the belt.
3. Clean concentrates are delivered near the left-hand end of the belt, having travelled the greatest distance before being washed off at the side.

Each of these three products is delivered into a separate hopper, and by a simple arrangement of sliding plates the exact points on the belt at which the delivery of the middle product is divided from those of the headings and tailings can be altered, so that the percentage of each product can be regulated.

The amount of ore treated in twenty-four hours by one of these machines is stated to be 4 to 5 tons if the concentrates are iron pyrites, and 3 to 4 tons if they are galena and blende.

The Wilfley Table is shown in Fig. 48. It consists of a smooth shaking surface, of which Fig. 49 is a diagrammatic plan. The light, worthless material is washed down sideways towards F G by a flowing sheet of water, while the heavy concentrates are thrown towards the upper end, C G, against gravity by a longitudinal jerking action. The pulp is supplied from the wooden feed box, A B, 5 feet 1½ inches long, through a number of small holes. Clear water is supplied from the trough, B C, which is 9 feet 10½ inches long, also through small holes. A sheet of linoleum is tacked down over the whole table and is fully exposed at D. A number (usually 46) of wooden riffles shown at E are placed on the top of the linoleum and are nailed down. The longest riffle runs the whole length of the table, and the shortest riffle is about as long as the feed box. The riffles are ¼ inch high at the "head end," A F, and taper down to nothing at the "concentrates end," C G. They are about ½ inch wide, the shorter ones being narrower, and are separated by ⅞-inch spaces so that the distance between centres is about 1⅜ inches. The fall from the back, A B C, to the front, F G, can be adjusted by wedges, and is usually about 1 inch in 13 inches. The rise from the "head end," A F, to the concentrates end, C G, is usually set at about ½ inch. The length of the side F G is 13 feet 9 inches, and the width at C G is 5 feet ¼ inch. The length of the perpendicular from A on F G is 6 feet 1 inch, and from A on C G is 15 feet 9½ inches. The angle at G is a right angle and that at F obtuse. The shaking mechanism is situated at the end A F, and consists of a movement towards C G, beginning slowly and uniformly accelerated by means of a toggle joint, and a return movement towards A F effected by a spring, beginning suddenly and uniformly retarded. The effect of this vanning motion is to a certain extent similar to that of a bump delivered at the end C G, the concentrates moving uphill towards G. The length of the stroke is about ¾ inch and the number about 240 per minute.

The tailings pass over the front F G, and the concentrates over the end C G. The middlings pass over near G, in the last 16 inches of the front, falling into a special launder, whence they are returned to the feed box or sent to other machines.

The size of the pulp treated on this table is usually from 16 to 30 mesh. The table does not save the finest slimes, and the Wilfley slime table, which consists of a series of moving troughs with canvas bottoms, is recommended in some cases for the treatment of the tailings from the ordinary Wilfley tables. The capacity of the Wilfley table is given as varying from 12½ to 30 tons per day, the amount of wash water required being also variable. The power required is about 1 H.P. per machine. It is stated that 10,000 of these machines are in use in various parts of the world, many of these being engaged in concentrating gold ores.

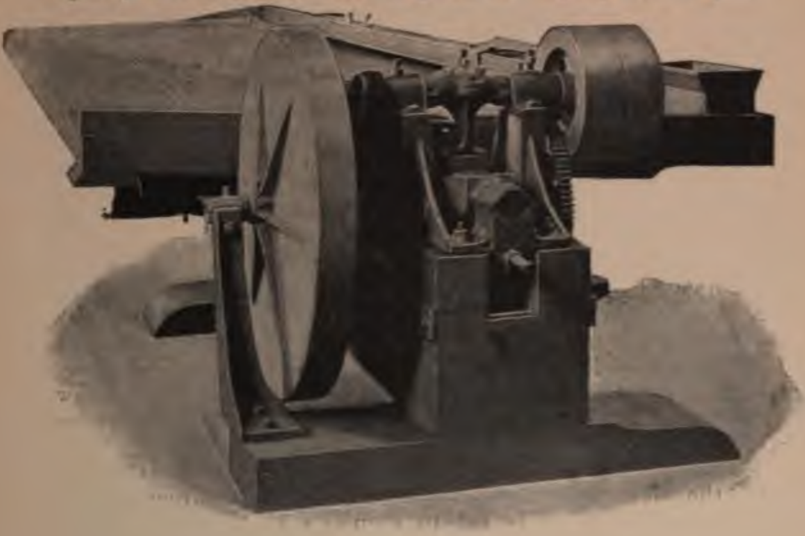

Fig. 48.—The Wilfley Table.

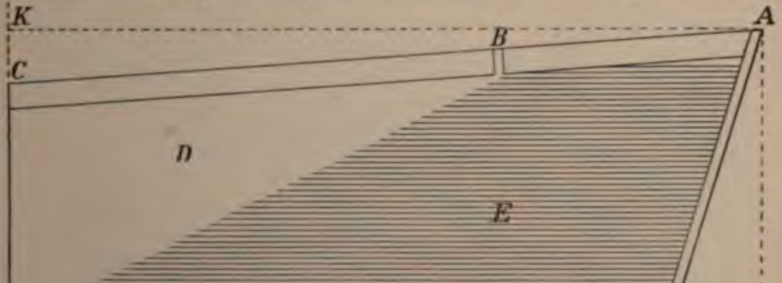

Fig. 49.

Hartz Jigs.—These machines differ in principle from all those previously described, inasmuch as the particles are separated by their fall through a somewhat deeper column of water than is the case on inclined tables, while a series of blows from below, causing waves moving upwards, continually brings the particles into suspension, and allows them to drop again. The initial period of the fall in water, during which the motion depends chiefly on density, is thus continually reproduced, and the result is a perfect separation of heavy from light particles of ore when working on any materials except the finest pulp. Jigs consist of sieves supporting beds of ore, which are completely immersed in water; the ore is raised and allowed to fall by a quick succession of currents of water caused by the sudden action of a piston below, which is so worked that the upward movement or pulsation resembles that produced by a blow, while the downward movement is gradual. Under these conditions the heavy particles work downwards and pass through the sieve, while the lighter gangue is carried away horizontally by a stream of water introduced either from below or from above. Such machines are especially suitable for coarse ores.

In the Hartz jig a layer of coarse heavy particles is spread on the sieve to prevent too much of the ore from passing through. The stuff is fed in regularly at the head of the jig, and the strokes of the piston raise both the bed of heavy particles and the ore. The heaviest grains of ore find their way during the downstroke into the interstices of the bed, gradually pass through it, and coming to the screen, fall through into the tank below. The lighter particles cannot descend, and are gradually washed over the end partition by the continuous supply of water. Two products are therefore given, neither requiring further treatment if the conditions are favourable, and the machine properly adjusted. The wire meshes of the screen are always much larger than the ore treated, and the bed is composed of material of as nearly as possible the same density as the concentrates to be obtained, and is usually from $\frac{1}{2}$ to 1 inch in depth. The number of strokes of the piston per minute is from 60 to 80 with coarse sand of $\frac{1}{12}$ inch in diameter, and 200, 300, or even 400 with very fine sand, approaching slimes. The length of stroke varies under the same conditions from $\frac{2}{5}$ to $\frac{1}{5}$ inch, and in the case of very fine, almost impalpable sand, the stroke may be diminished till it becomes a mere tremor. Some of the highest authorities on concentration have stated their belief that for enriching even very fine sand, the Hartz jig is the simplest and most economical machine yet invented, and requires the least amount of labour.*

Concentration by Oil.—It has been found that in the case of certain ores, residuum oil separates sulphides and metallic particles from earthy or stony materials without reference to their respective densities. In the Elmore concentrating process, which was introduced at Glasdir, near Dolgelley, in Wales,* the ore is crushed in Huntington Mills, and the pulp mixed with oil in a revolving drum. The mixture is discharged into circular vats where the water and sand subside, leaving the oil carrying the pyrites floating on the top. The oil is then pumped into a "hydro-extractor," which is run at 750 revolutions per minute, and the oil separated from the ore. The amount of oil added is equal to the weight of the ore, and the loss of oil is about two gallons per ton. The oil residuum used has a density of about 0·9, and is made more viscous, if necessary, by the addition of paraffin wax or similar materials. The oil will carry from 10 to 20 per cent. of mineral when it is fully charged. The cause of the selective action exerted by the oil is not known. It is suggested by H. F. Collins † that a smooth lustrous non-absorbent particle is wetted by the oil, and adheres to it, whereas the oil will not adhere to a rough porous particle when it is soaked with water. He found that an ore containing copper and iron pyrites, and consisting mainly of crystallised minerals, wollastonite being the most abundant constituent, did not lend itself readily to oil concentration. At Glasdir, the average saving was 69 per cent. of the gold, 65 per cent. of the silver, and 70 per cent. of the copper, with a rate of concentration of about 14 to 1.

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* Callon's *Lectures on Mining*. Eng. edition, vol. iii., p. 103.

Concentration by air instead of water is effected in a number of machines, which are used in certain cases. In hot dry climates, where water is scarce, where the ore is readily made quite dry, or in the event of water exercising a harmful chemical action on the ore, air concentration may be sometimes usefully employed. The jigging principle is used in Krom's pneumatic jig, centrifugal force in Clark & Stansfield's machine, and a steady current of air in Edison's dry blower (see p. 58).

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* *Trans. Inst. of Mng. and Met.*, vol. viii. (1900), p. 379.

† *Ibid.*, p. 394.

CHAPTER X.

DRY CRUSHING.

Introduction.—In those cases in which gold ores are treated by crushing and amalgamation, and the whole of the tailings, or only the concentrates obtained from them, are subsequently treated by the cyanide process or by chlorination, the method of crushing will be determined by the considerations discussed in the chapters on Amalgamation, Concentration, and Re-grinding, and will depend partly on the state of aggregation of the free gold. If, on the other hand, ores are to be treated in the first instance by "wet processes," special regard may be paid to the method of crushing as affecting the suitability of the crushed product for leaching. When the ore is to be roasted before treatment by solvents, as is invariably the case in chlorination and sometimes in cyaniding, dry crushing is always resorted to. It is also sometimes used when the ore is not roasted. The method is especially valuable if water is scarce.

In dry crushing there are two somewhat opposite conditions to be fulfilled, viz.:—

1. The crushed product should be fine enough to admit of the whole of the gold being laid open to the attack of the solvent.
2. It must be coarse enough to allow all the soluble compounds of gold thus formed to be washed out of the ore rapidly and easily.

It is quite impossible to fulfil completely either of these conditions on a large scale, although in the laboratory some ores, when treated with the greatest care and patience, may be made to yield the whole of their gold. In practice it is better to aim at an extraction of from 85 to 95 per cent. (98 to 99 per cent. being attained in some rare cases), and to do this at a low cost. With this object in view, the ore is kept coarse and is usually passed through screens with only 8 to 30 holes to the linear inch. One of the points to be attended to is the attainment of uniformity in the product, as any considerable proportion of slimes enormously increases the difficulties of ordinary leaching. Now that the separation of the slimes from dry crushed ore for separate treatment can be effected by air, this point has become of less importance. Dry crushing is usually effected by stamps, by rolls, or by ball-mills. The ore is previously passed through rock-breakers and then dried before it is finely crushed.

Drying.—Most ores must be dried before they can be finely crushed by means of rolls or other dry crushing plant. The screens soon become clogged by damp ore, especially if it is argillaceous. The moisture must be reduced to 1 or 2 per cent., and Argall points out* that the removal of water of hydration by heating to about 250° makes it much easier to screen hydrated ores.

The oldest method of drying was to spread the ore, after the large lumps had been removed by a grizzly and crushed to 1½-inch size, on large flat areas heated from below by flues from the roasting furnace. The floor was usually covered with iron plates. After being dried, the ore was shovelled up and passed to the crushing mill. This plan involved much additional handling of the ore, and was a source of ill-health among labourers, besides requiring great floor space. It has been superseded by the adoption of inclined continuous-discharge, revolving iron cylinders, similar to the Howell-White furnace, but not lined with bricks. The ore is passed through these, and is dried by the products of combustion of a fire, which are also passed through it. One such cylinder, of 3 feet in diameter and 18 feet long, will dry from 30 to 40 tons of ore per day, at a small cost for fuel and power.

Argall† uses a tubular drier, consisting of a nest of four or six tubes similar to his roasting furnace (*q.v.*), but of lighter construction. He gives the capacity at from 80 to 300 tons of ore per day, the consumption of coal at about 30 lbs. per ton of ore, and the cost of drying ore containing 6 per cent. of moisture at 5 cents per ton in Colorado.

An alternative furnace—viz., Stetefeldt's shelf drying kiln—was described in a paper read by the inventor at the meeting of the American Institute of Mining Engineers, held at Roanoke, Virginia, in June, 1883. In principle it resembles the Hasenclever furnace, a number of shelves being arranged zig-zag above each other in a vertical shaft, down which the ore slides, falling from shelf to shelf, while the products of combustion from a furnace rise through it. It is 21 feet high, and dries from 30 to 50 tons of ore per day. Although the first cost of this furnace is considerable, its working expenses are said to be low.

Stamps.—When stamps are used for dry crushing, double discharge mortars are provided (see Fig. 50), and the screens are put low down, giving a small depth of discharge. These arrangements are necessitated by the difficulty of discharging the crushed ore from the mortar, the only means of doing this being the dashing of the dry ore against the screens, due to the fall of the stamps. Expediting the delivery of the ore by a blast of air forced into the mortar, or by suction from outside, is not often attempted. The result of the slowness of discharge is that with a given fineness of the mesh of the screens, dry crushing gives a finer product than wet crushing. This is an advantage with many ores, in respect that the gold is more readily accessible to the solutions, but, on the other hand, the product is more difficult to leach. The output in dry crushing is much lower than in wet crushing by stamps, and the cost per ton of ore is stated to be from 50 to 100 per cent. more.

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* *Trans. Inst. Mng. and Met.*, vol. x. (1902), p. 241.

† *Loc. cit.*

One of the difficulties in dry crushing is the production and loss of dust, which is generally richer than the bulk of the ore, for the reason that the sulphides are more brittle than the gangue. This difficulty is partly met by surrounding the crushing machine with an outer casing, and drawing off the escaping dust with air suction. Nevertheless, some dust escapes, involving a loss of values and also some injury to the machinery and to the health of the workmen.

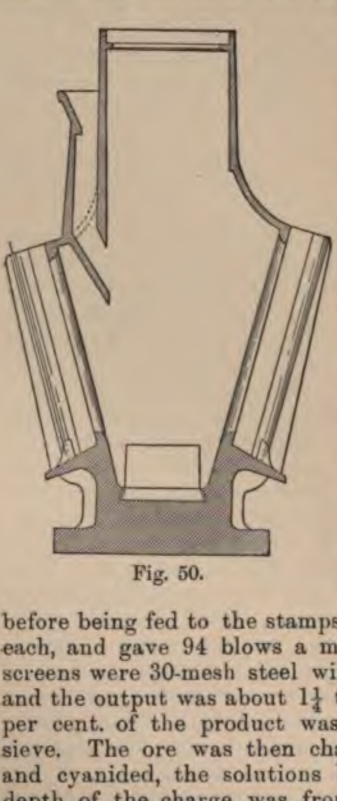
Fig. 50.

Dry crushing by stamps is largely used in the treatment of silver ores by pan-amalgamation or by hyposulphite (thiosulphate) leaching. It was introduced on a large scale in 1894 for crushing the oxidised quartzose ores from the upper levels of the Waihi and other Ohinemuri (Upper Thames) Mines, New Zealand, and was finally abandoned in favour of wet crushing in 1904, partly owing to the fact that the character of the ore had changed with the working-out of the upper levels.

At the Waihi Mill* the ore was dried and partly roasted in kilns, with wood as fuel, and then broken down in rock-breakers before being fed to the stamps. The stamps weighed 1,000 lbs. each, and gave 94 blows a minute with a 6½-inch drop. The screens were 30-mesh steel wire (900 holes to the square inch), and the output was about 1¼ tons per stamp per day. Over 77 per cent. of the product was fine enough to pass a 90-mesh sieve. The ore was then charged into shallow leaching vats and cyanided, the solutions being run in from below. The depth of the charge was from 20 to 30 inches, and vacuum leaching was used. A depth of three feet of ungraded pulp was found to be unmanageable, and no attempts were made to separate the slimes. According to James,* the cost at the Waihi Mill was stated to be 9·58d. per ton less for dry stamping than for wet stamping on the same ores.

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* J. M'Connell, *Trans. Inst. of Mng. and Met.*, vol. vii. (1899), p. 26. F. M. Merricks, *Idem*, p. 35. P. Morgan, *Trans. Australasian Inst. of Mng. Eng.*, vol. ix., Part II. ; *Eng. and Mng. Journ.*, Aug. 4, 1904, p. 183

Rolls.—It is generally considered that rolls are specially adapted for intermediate crushing, taking a product from rock-breakers with a maximum diameter of from ¾ inch to 1½ inches, and reducing it to a size passing through a screen of 12 or 16 meshes to the square inch. Finer crushing, however, is often carried out. The exact limit at which rolls cease to be economical machines is still a matter of doubt. Like other dry crushing machines, they are better suited for soft friable material than for either hard or caking (clayey) ores.

Rolls are cylinders, between two of which pieces of ore are drawn and crushed by compression. They are placed with their faces a short distance apart, this distance varying with the degree of fineness to which the ore must be reduced. The main driving power is applied to the shaft of only one roll, by means of belt pulleys, the other roll being driven only with sufficient force to ensure that the rollers will always take hold of the ore, and also to keep them in motion when no ore is passing between them.

In the older forms, tooth-gearing was used instead of belt pulleys for the application of the power, and the two rolls were compelled to revolve at an equal rate of speed by gear-wheels, connecting together, placed on the axles. The advantages of belted rolls are, that a higher speed can be easily attained, and also that if the rolls were to become jammed from any cause, the belts would slip or be thrown off, while the tooth-gearing would be broken. Geared rolls, however, are still largely used for coarse crushing. The rolls have crushing tires made of steel or of chilled iron. Chilled iron is much cheaper, but the wearing of the faces is more rapid and less uniform than in the case of steel rolls. Emery wheels are used for levelling the unevenly worn faces of the rolls. The crushing tires can be taken off and replaced when they are worn out. In some rolls the crushing strain is taken up by powerful springs, which press the rolls towards one another; when particularly hard fragments are passing through the rolls, they are forced apart against the action of the springs. It is desirable, in order to keep the wear of the faces even, that the rolls should always be kept parallel, and special appliances are used to ensure this. The hopper is designed to spread the ore evenly across the crushing face, and the rolls, screens, elevators, &c., are all securely boxed in with a wooden housing. This last precaution is necessary in order to prevent loss by floating dust, which otherwise may be large, the richest part of the ore thus passing off, and not only making the atmosphere of the mill insupportable, but having a disastrous effect on the bearings of the machinery. Rolls are usually from 12 to 16 inches across the face, and from 22 to 36 inches in diameter.

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* *Cyanide Practice*, 1st edition (1901), p. 104.

Richards points out[*] that ore may be crushed by rolls in two ways, according to the rate of feed, speed of rolls, &c. If the speed is high and the feed light, each particle of ore is crushed separately between the rolls. In this case, which he calls "free" crushing, there will be a maximum of coarse and a minimum of fine material produced. In "choke" crushing, the ore is fed in a thick stream between the rolls, so that the particles crush each other and a maximum of fine material is produced. Part of the power will be used in compressing the loosely-packed stream. According to Richards, free crushing is the more advantageous course, provided that a very fine product is not required.

The prevailing opinion seems to be that rolls are not economical fine crushers. A certain percentage of the material passing through rolls is crushed very fine, and if the whole product is to be fine, the remainder is sieved out and returned to the rolls. A further quantity, smaller than before, will be sufficiently reduced by the second passage through the rolls. According to Fischer Wilkinson,[†] if fresh coarse ore, equal in quantity to the separated fines, is added to the intermediate product, it is probable that the output will remain constant, and that the power required will be less than in the stamp battery.

Rolls are not generally used to crush finer than about 20 mesh, but Edison proposes to crush to 200 mesh, using choke crushing and corrugated rolls.[‡] Corrugated rolls were formerly tried at Mount Morgan and elsewhere, but were abandoned as being of little value, wearing unevenly, and soon getting out of order. Edison's process appears to necessitate a reduction of comparatively coarse material to excessively fine particles in a single pair of rolls. In general, however, it is considered that gradual reduction in successive pairs of rolls is more economical, the product of one pair of rolls passing to another set slightly closer together. An application of this principle is given in the following description[§] of the Mount Morgan plant, Queensland, formerly in use :—

At Mount Morgan the ore was soft and friable (the "Gossan Cap"), with about 10 per cent. of hard boulders of quartz. It was broken in Krom jaw-breakers (hinge at bottom) to $\frac{3}{4}$-inch gauge, dried, and then passed successively through four pairs of belted rolls, all running at 112 revolutions per minute. The distances between the faces were $\frac{3}{8}$ inch, $\frac{3}{16}$ inch, $\frac{1}{16}$ inch, and 0 respectively, the first two rolls being 26 inches in diameter and 15 inches wide, and the last two 36 inches in diameter and 16 inches wide. Revolving hexagonal screens or trommels were used after each pair of rolls, the brass screens having 20 holes to the linear inch. The first trommel was protected inside by a steel screen with 16 holes to the square inch. The coarse product was in each case sent to the next pair of rolls. The roll tires were of cast steel, 3 inches thick, and required turning up again in from one to three months. They lasted twelve months, and were discarded when worn down to ½ to 1 inch thick. The wear was 0·108 lb. of steel per ton of ore crushed. The power required was 0·8 I.H.P. per ton crushed in 24 hours. The capacity was 62½ tons per day for each set of 8 rolls, and the total cost, exclusive of lighting, breaking, and drying, was 3s. 10d. per ton. The crushed ore was roasted and chlorinated (see Chap. xv.). The rolls have now been superseded by Krupp ball mills (*q.v.*).

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[*] *Ore Dressing*, p. 98.

[†] *Inst. of Mng. and Met.*, discussion, Feb. 16, 1905, p. 30.

[‡] "Dry Crushing," by Messrs. Simpkin & Ballantyne. *Idem*, Oct. 20, 1904.

[§] N. F. White, *Trans. Australasian Inst. of Mng. Eng.*, vol. vi. (1900), p. 36.

"At the Bertrand Mining Coy.'s Lixiviation Mill, Nevada, where Krom's rolls are used for pulverising silver ores with a quartz gangue, 15,000 tons of ore were crushed before it was found necessary to put new tires on the finishing rolls; while after a further crushing of 5,000 tons of ore, the tires of the roughing rolls were still expected to be good for two or three month's work. In neither case were the tires found to be at all grooved, the reason for their renewal being that they had become worn too thin for further work. The rolls in the Bertrand Mill are stated to crush 50 tons of hard quartzose ore in twelve hours to pass through a 16-mesh screen; while in the Mount Cory Mill, Nevada, 50 tons are reduced to 30-mesh in the same time."* In these mills silver ores were crushed for treatment by roasting and lixiviation with a hyposulphite solution.

Comparison between Rolls and Stamps.—As the subsequent treatment of an ore determines its method of crushing, no accurate general comparison of stamps and rolls can be made. A comparison is only possible in the special cases where both methods of crushing are applicable. Wet crushing by rolls need not be considered, as it is not practised; even where the advocates of rolls wish to replace stamps by rolls for wet crushing and amalgamation, they propose that the ore should be crushed dry and then wetted down. Dry crushing by stamps is usually about one-third more expensive than wet crushing, as the capacity falls off to that extent, in spite of double discharge, assisted occasionally by currents of air given by blowers, which are designed to carry the crushed ore against the screens. The amount of slimes made is also large. Advantages in the use of rolls for dry crushing have been stated to be

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* Curtis on "Gold Quartz Reduction." *Proc. Inst. Civil Eng.*, 1892.

"the fewness of the wearing parts, and consequent small cost of repairs; the great efficiency of the process, in that the ore escapes from the rolls immediately it is crushed, so that over-crushing is unlikely to occur, and the great capacity of rolls, effective work being constantly done, and the amount of crushing surface brought into contact with the ore per minute being very large. The prime cost of the rolls is considerably less than that of stamps of the same capacity."*

The capacity of rolls has perhaps been frequently over-estimated owing to the assumption having been made, that the product of equal crushing surfaces must be the same. Thus, Curtis observes in the paper already cited:—"As an index of the capacity of Krom's rolls, it may be stated that two sets of 26-inch (diameter) rolls, with faces 15 inches long, give rather more effective crushing surface than fifty gravitation stamps, each 8 inches in diameter, falling at the rate of ninety drops per minute. In making this calculation, the average diameter of the rolls is taken as only 24 inches, so as to allow for their gradual wearing, while their speed is taken at 100 revolutions per minute."

Although the calculated crushing surface is as stated, it does not follow that the capacity of the two sets of machinery compared with one another is the same, since it will depend on the pressure as well as the crushing surface. No doubt the pressure at the moment of impact of a 900-lb. stamp is enormously greater than that exercised by the faces of the rolls.

On this point J. Richards, M.E., writes:†—"The coefficient of (crushing) effect in such machines (as revolving rollers and reciprocating machines) is as the area of the acting surfaces, and the speed with which they approach each other. The mistake in respect of the crushing power of rollers comes from confounding their circumferential velocity with the working one—that is, with the parallel velocity at which the surfaces approach and leave each other," that is, the velocity at right angles to the crushing surface. Richards elsewhere shows that a Blake machine in this way actually outruns a revolving roller, although its crushing face travels more slowly than the periphery of the roll. He proceeds:—"The same principle holds good in respect to stamps, the crushing surfaces having a parallel approach, while with rotary machinery the approach is not parallel, except on an imaginary line at the centre." As a matter of fact, the product of two pairs of rolls amounts to only about 40 to 50 per cent. of that which would be given by a fifty-stamp mill on the same ore. In the absence of exact comparative data regarding the same ore, no precise figures regarding the relative capacities can be given.

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* Curtis, *loc. cit.*

† *Prod. of Gold and Silver in the United States*, 1880, p. 369.

A point to be noticed is that, in machines which act by pressure applied on the principle of the lever, such as reciprocating-jaw crushers or rolls, the whole force necessary to crush the lumps of quartz is transmitted to the fulcrum, this fulcrum being represented in the case of rolls by the bearing surface at the axle. The consequence is that the frame must be made very strong and heavy, and the axle bearings attended to with great care, if rapid wear is to be avoided.

Ball Mills.—Mills in which ore is crushed by iron balls have long been in use. In the earlier forms the balls travelled round in a slot which they fitted loosely, the crushing being effected partly by centrifugal force. The wear of the balls soon interfered with the efficiency of some of these machines, and none proved very successful. In later machines, a number of small balls enclosed in a rotating drum crush the ore by falling from little shelves, and this type has passed into extensive use for dry crushing in Australia and America.

The Krupp or *Krupp Grüsonwerk* Ball Mill has been highly successful. It consists of a cylindrical drum rotated by belting, but geared down by toothed wheels (see Fig. 51). The interior of the drum consists of from 5 to 10 overlapping grinding steel plates, *a*, *b*, so arranged that the balls drop a few inches in passing from each plate to the next. The half, *a*, of the grinding plates is perforated with ½-inch tapered holes, and the other or main grinding half, *b*, is bent inwards towards the axis to raise the balls above the next succeeding plate. The crushed ore passes through the perforations in *a*, and falls upon a coarse sieve, J, constructed of finely perforated steel plates, which protect the outer fine sieves, K, of phosphor bronze or steel wire gauze. The coarse particles retained on both inner and outer sieves, which enclose the crushing drum and rotate with it, are returned by elevator plates, *l*, *m*, to the interior of the drum through perforated plates, *n*, guarding apertures between the grinding plates. The material, which need not be smaller than 2 or 3 inches in diameter and is sometimes larger, is fed into the drum from a hopper at the side. The balls are of chilled cast iron or cast steel and of various sizes, the largest being about 5 inches in diameter. As they wear down, others are added. The material which passes through the fine sieves falls into a hopper which is continuous with the dust-proof casing of sheet iron. The interior of the sides of the drum is lined with chilled iron or steel plates. The mills are made in eleven sizes, the largest having a drum 8 feet 10 inches in diameter and 4 feet 6 inches in width. This size rotates twenty-one times per minute and contains about 2 tons of steel balls. The power required is approximately 30 H.P., and the amount crushed is about 3 tons of quartzose ore per hour through a 40-mesh sieve. Smaller mills, however, are

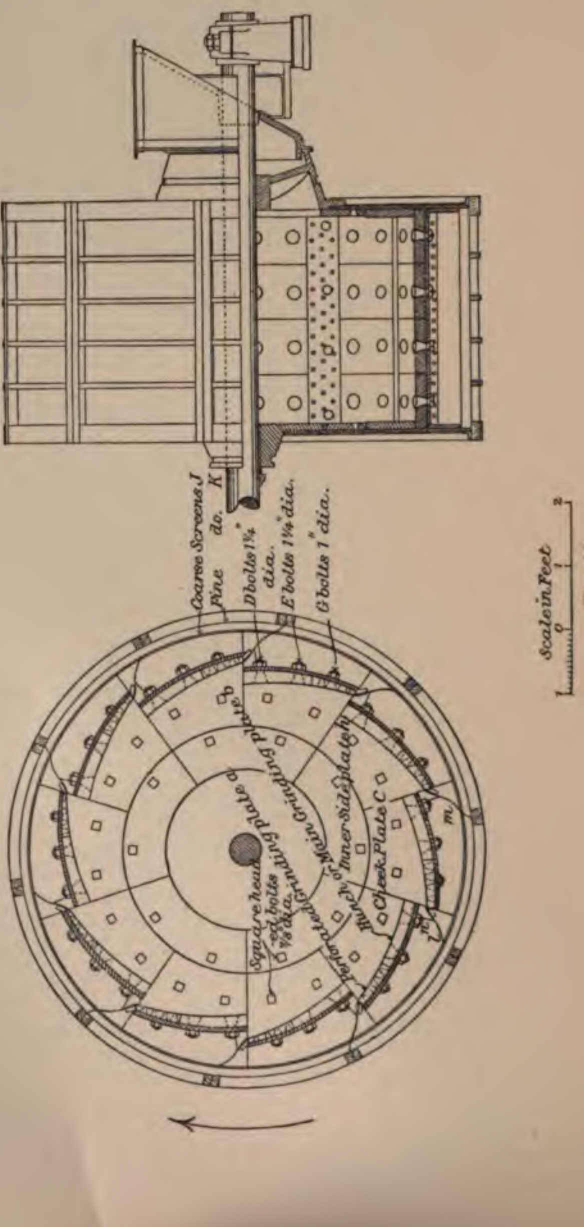
Coarse Screens J
Fine do. K
D bolts 1¼" dia.
E bolts 1¼" dia.
G bolts 1" dia.
Square head-ed bolts ⅝" dia.
Grinding plate a
Grinding plate b
Main Grinding plate
Perforated Inner Side plate h
Check Plate C
m
n
Scale in Feet
1
0
1
2

Fig. 51.

usually employed, especially the No. 5 mill (see below) which contains 20 to 22 cwts. of steel balls and has an output of about 1½ tons of Kalgoorlie ore per hour, using a 20-mesh screen. One steel ball weighing 18 lbs. is added each day to compensate for wear.[*]

The same mill is made for wet crushing with an increase of output of about 30 per cent. Sprays of water are introduced inside the drum and are directed against the sieves, and the lower part of the dust casing is converted into a spitzkasten, which separates slimes from sand. The wet crushing mill, however, has not yet passed into regular use.

The Krupp ball mills installed at Mount Morgan,[†] where they were for the first time used for crushing gold ores, are shown in Fig. 51, which is reproduced by the kind permission of the Australasian Institute of Mining Engineers. They are No. 5 mills, 7 feet 6 inches in diameter and 3 feet 9 inches wide, and containing about a ton of steel balls. There are 10 cast-steel grinding plates, and the fine screen has 400 holes to the square inch. The mills were driven at from 21 to 22 revolutions per minute, and crushed soft friable ores containing 10 per cent. of hard-quartz boulders (see above, p. 192). The ores were delivered from rock breakers. Sixteen mills crushed about 123 tons in eight hours, so that the product of each mill was 23 tons per day of 24 hours. The power required was 13 I.H.P. per mill. The wear amounted to 0·919 lb. of steel plates, bolts, &c., and 0·725 lb. of steel balls per ton of ore on a run of 12 months, ending 31st May, 1899. The total cost for breaking, drying, and crushing was 3s. 11d. per ton against 6s. 9d. per ton when Krom rolls were used. Exclusive of power, the cost of crushing in the ball mills only was 1s. 3d. per ton.

These mills have also been largely used in crushing at Kalgoorlie, where the experience is similar to that at Mount Morgan. The screens used at the Great Boulder Main Reef[‡] were 900 to the square inch, and 60 per cent. of the product passed a 90-mesh screen. The cost of crushing, exclusive of power, was 1s. 6d. per ton.

The Griffin Mill[§] is also used at Kalgoorlie and elsewhere. It consists of a single roller about 18 inches in diameter with vertical axis. It is suspended from above, and made to rotate about 200 times a minute. At the same time it rolls round on the inside of a die ring 30 inches in diameter, so that it has a gyratory motion. The screens are placed above the die ring.

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[*] Evan Simpson, *Trans. Inst. Mng. and Met.*, Oct. 15, 1903.

[†] "Some Notes on Dry Crushing," by N. F. White, *Aust. Inst. Mng. Eng.*, vol. vi. (1900), p. 37.

[‡] A. G. Charleton, *Gold Mining and Milling in Western Australia* (1903), p. 247.

[§] See above, p. 161.

The machine is also used for wet crushing. It produces a finer product than ball mills when using an equivalent screen, and is thus more suitable when the ore is to be slimed than when the product is to be treated by leaching without separation of the slimes. James gives the cost of crushing through a 15-mesh screen at Kalgoorlie as 1s. 10d. per ton, exclusive of power.*

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* *Cyanide Practice* (1901), p. 111.

CHAPTER XI.

RE-GRINDING.

Discussion of Coarse and Fine Crushing.—The general system of treating "free-milling" gold ores consists in crushing in the stamp battery, amalgamating either in or outside the battery, and treating the tailings by solution processes. The free or amalgamable gold is caught on amalgamated plates, and the gold contained in pyrites, tellurides, &c., is left for the attack of cyanide or of chlorine. Ores are of infinite variation, however, and the percentage of gold recoverable by amalgamation can hardly be fixed even for a single ore. It depends on many factors, the main one being in most cases the degree of fineness to which the ore is crushed. In general some of the gold is distributed through the ore in an extremely fine state of division, and the finer the crushing the more gold is laid open to the attack either of mercury, cyanide, or chlorine. This consideration points to the desirability of reducing to "grits" or "sliming" all the ore, if it can be done cheaply enough.

On the other hand, the values are generally in part contained in sulphides, tellurides, &c. These form the richest part of most ores, and retain their gold with greater obstinacy than the other constituents of the ore. They are also the most brittle, so that if the whole ore is reduced to a fine state of division the sulphides, &c., are converted into an impalpable slime. In this condition they cannot readily be saved by concentration. Until recently another objection to fine crushing was that the crushed material could not be treated by cyanide or chlorine on account of the mechanical difficulties of leaching. It was necessary in chlorination to use shallow leaching vats, and in cyaniding to separate the slimes * from the sands, and to leave the former untreated. Now that slimes can be treated by decantation or filter pressing, the objection to their formation has less force.

Before the introduction of the cyanide process, when the general methods of treatment were crushing and amalgamation, followed by concentration and the treatment of the concentrates by roasting and chlorination, the aim was to avoid the formation of slimes. At that time the value of crushers giving a uniform product seemed especially great, and rolls were strongly advocated as against stamps, which produced more fine material. Later, when the Kalgoorlie ores began to be treated by dry crushing and roasting, the uniform product of ball mills was found to be better than the partially-slimed product of Griffin mills, because of the greater difficulty of roasting the latter.* When ores are subsequently to be roasted there is little advantage in fine crushing, because the particles usually become porous, so that chlorine or cyanide can penetrate into their interior and dissolve gold which has not been exposed on the surface of the grains.

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* An exact definition of slimes is desirable. Perhaps the most useful definition is "that portion of the crushed ore which, owing to its physical condition, cannot be leached by percolation under the action of gravity." Such material does not subside readily in water. In this chapter finely-crushed material which can be leached is called "grit."

Now, however, it is being realised more and more that, in the case of ore which is treated without roasting, the finer the state of division the higher the percentage of gold that can be extracted. The method of re-grinding the product of the stamp battery or of dry crushers in other machines has lately been developed, resulting in a higher percentage of extraction in many cases. This has been followed by attempts to increase output by using coarser battery screens, and also to use a method of gradual reduction in successive machines, rather than one of great reduction in size in a single machine.

It has already been pointed out that if ore is retained for a long time in the mortar of a stamp battery by increasing the depth of discharge and the fineness of the screens, and by diminishing the screen area, a higher percentage of gold can be amalgamated in the battery and the fineness of the product increased. The result is that only 1 or 2 tons of ore are crushed per stamp per diem, but most of the gold is caught by the inside copper plates. When the screens are coarse and set low down, the crushed ore is rapidly discharged, less grit and slimes are made, and the percentage of gold amalgamated in the mortar is small, so that the advantage of adding mercury with the ore falls off and there is a tendency towards complete discontinuance of the practice, leaving all gold-catching by amalgamation to outside plates. This is a method similar to the old German practice (see above, p. 98), and the view seems to be spreading that this reversion to ancient methods is desirable, and that the stamp battery is neither the most economical fine crusher nor the most efficient amalgamator.

The influence of the screen on the output of ore is illustrated by the following results obtained in 1904 at the Glen Deep Mill on the Witwatersrand:—

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* "Dry Crushing and Roasting at Kalgoorlie," by Evan Simpson, *Inst. of Mng. and Met.*, Oct. 15, 1903.

Screen, holes per square inch.	Output per stamp in twenty-four hours.	
800	4·9 tons.	
200	6·9 ,,	
150	7·8 ,,	
100	8·04 ,,	
64	9·4 ,,	10·68 tons under the most favourable circumstances.

With the stamp battery relegated to its former position of a crusher only, the claims of the steam-stamp are strengthened, and something may again be heard of it in connection with gold milling.

Re-grinding in Pans.—Re-grinding of coarsely-crushed ore is effected in pans or in tube mills, which are described below. An account of amalgamating pans has been given above (p. 164). Wheeler pans have been found advantageous at the Ivanhoe Mill * for re-grinding, and Broadbridge suggests that the pan should be used as an intermediate grinding medium, afterwards passing the product through tube mills.

At the Ivanhoe Mill it was found by Nicholson † that the product of 10 stamps could be dealt with by two pans, using battery screens of 10 mesh or 100 holes to the square inch. The screen tests gave the following results:—

	Pulp Leaving Mortar Boxes.	Product of Spitzkasten Entering Pans.	Product Leaving Pans.
	Per Cent.	Per Cent.	Per Cent.
On 20 mesh, .	15·6	26·3	Nil
,, 30 ,, .	9·7	15·7	,,
,, 40 ,, .	4·2	6·4	,,
,, 60 ,, .	16·4	16·4	10·6
,, 90 ,, .	8·0	8·7	18·6
Through 90 ,, .	46·1	26·5	70·8

Nicholson introduced the use of compensating weights to fit on the muller when it had become partly worn. Even when these are used, the product of pans is not quite so fine as that given by tube mills, but there are compensating advantages. Nicholson considered that five 5-foot pans would do the same work as two 16-foot tube mills (so that the first cost of pans would be less than half that of tube mills), and that the cost of running would be about the same. The power required per pan was about 5·3 H.P. at the Ivanhoe Mill. G. A. and H. S. Denny ‡ give as other advantages in favour of pans—

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* W. Broadbridge, discussion at meeting of *Inst. of Mng. and Met.*, Nov. 17, 1904.

† *Idem.*

‡ *Mining Magazine*, Sept. 1905, p. 180.

(1) Their accessibility.
(2) The possibility of internal amalgamation.
(3) The smaller capacity of the unit.
(4) The saving in time of renewals of working parts.

Nevertheless, tube mills seem to be generally preferred to pans, partly because the former yield a finer product.

Tube Mills.—These have been used for over 20 years in ore treatment in Cornwall, and are also used in the cement industry for dry grinding. They were first applied to gold ores in West Australia in 1899, and are used wet. They consist of revolving cylinders, the interior of which is perfectly plain. They contain a number of hard balls. They differ from ball mills (*q.v.*) essentially in having the inlet for ore at one end and the outlet at the other, and in the absence of provision in the machine itself for the return of uncrushed material (oversize). It is therefore necessary for the cylinders to be of greater length than in ball mills. The number of balls used is also much larger, and, according to one view, the reduction of the ore is effected by grinding rather than by impact, as in ball mills, so that this constitutes another essential difference.

The *cylinders* are from 3½ to 6 feet in diameter, and from 13 to 26 feet long. They consist of welded steel with cast-iron or steel heads.

The *lining* consists of hard wood, chilled cast-iron, hard homogeneous iron, steel, manganese steel, quartzite, or flint (silex). Of these, wooden blocks were stated to be worn out in three days in South Africa, and cast-iron plates in three weeks, but in this case coarse ore was said to have been treated. Manganese steel linings lasted fifteen months at Kalgoorlie, and chilled iron liners from seven to ten months.* The great cost of manganese steel made it less economical than hard iron.†

The quartzite lining of the Davidsen Mill consists of blocks of quartzite, 8 inches by 4 inches and 2 inches thick, cemented in place. These linings are supposed to last for two or three years or even longer. The wear of flint linings is equal to the wear of the flint balls according to Dr. Diehl, but is far less according to S. Robinson.‡ In the Smidth-Davidsen mills the linings are not bolted through the cylinder, but in other mills bolts are used.

The *balls* usually consist of flint, although the use of steel balls has been advocated. The size varies up to 3 or 4 inches in diameter. According to M. Davidsen,§ in wet crushing large pebbles wear less rapidly than small pebbles and crush as finely.

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* Alfred James, *idem*, Oct. 20, 1904.

† Trewartha-James, *idem*, Nov. 17, 1904.

‡ Discussion on "Finer Crushing of Banket Ore," Inst. of M. & M., Nov. 12, 1904.

§ *Idem*, Dec. 15, 1904.

The larger the pebbles the coarser may be the ore fed in. For dry grinding small pebbles of 1 to 2 inches in diameter are preferable. The Greenland flints, found on the Danish shores, are more durable than the chalk flints of the French coast. The mill is best kept about half filled with pebbles, the weight of pebbles added to a mill being 44 lbs. per cubic foot of interior volume. The consumption of flints is about 1 lb. per ton of ore treated.

The *speed of revolution* of tube mills varies with the diameter. Krupp uses the formula $\frac{32}{\sqrt{D}}$ for the correct number of revolutions per minute, where D = internal diameter in metres. The Davidsen formula is $\frac{200}{\sqrt{D}}$, where D is the diameter in inches. These two formulæ are almost identical and would give about 26 revolutions per minute for a mill of 5 feet in diameter. In practice a somewhat higher rate of speed is used, amounting generally to 2 or 3 revolutions per minute more than corresponds to the formulæ given above.

The *power* required, according to Davidsen, is given by the formula $P = 0{\cdot}15\ N$, where P is the number of H.P. and N the internal capacity of the mill in cubic feet. According to James, the power required is less for wet grinding than for dry grinding.

The *capacity* of tube mills depends on the size of the material supplied to them, and on the fineness of the product. The amount of grinding done and the fineness of the product depend on the rate of feed and the length of the tube, as well as on the hardness of the ore. According to Gruessner,[*] at the Hannan's Star Mill, Kalgoorlie, the ore from two No. 5 dry ball mills was treated wet in a tube mill 16 feet 5 inches long and 3 feet 11 inches in diameter. The ball mills crushed 78 tons per day and, after classification, the coarser part of this, amounting to 38 tons, was ground to about 250 mesh in the tube mill. The pulp fed into the mill contained: coarser than 40 mesh, 20·5 per cent.; 40 to 60 mesh, 43·8 per cent.; 60 to 100 mesh, 28·1 per cent.; 100 to 150 mesh, 7·6 per cent. The product was not examined,[†] but returned to the spitzkasten and re-classified, the sands returning to the mill. The overflow (final product) contained: coarser than 100 mesh, 0·7 per cent.; 100 to 150 mesh, 4 per cent.; finer than 150 mesh, 95·3 per cent. The power consumption of the 16-foot mill was 30 H.P. The cost of re-grinding per ton of crude ore was as follows:—

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[*] *Idem*, Oct. 20, 1904.

[†] According to W. H. Trewartha-James (I.M.M., Nov. 17, 1904), the amount of oversize left after one passage through the mill was so great that some 268 tons of sands were passed through the mill per day for a product of 38 tons of slimes.

Power, ...	11·53d.
Flints and liners, . .	1·85d.
Labour, ...	3·50d.
Repairs, ...	0·99d.
Total, . .	1s. 5·87d., or 3s. 0·68d. per ton of material actually slimed in the tube mill.

The mill used was apparently one with central discharge.

At the Oroya Brownhill Plant, Kalgoorlie, according to W. Broadbridge,[*] about 7,500 tons per month were treated in six flint mills (five 13 feet 7 inches by 3 feet 8 inches, and one 12 feet 11 inches by 4 feet 1 inch), during the period February to July, 1904. The average amount of sands slimed per mill per hour was 1·77 tons, and the cost 1s. 9·33d. per ton treated.

The effect of re-fitting a tube mill at the Ivanhoe Mine was also given by W. Broadbridge as follows[†]:—

	Sands before Grinding.	Sands after Grinding.	
		16-ft. Mill fitted with Old Liners and Small Flints.	Same Mill fitted with New Liners and Large Flints.
	Per Cent.	Per Cent.	Per Cent.
On 40 mesh, .	38·4	0·5	0·10
,, 60 ,, .	41·6	14·0	6·28
,, 100 ,, .	15·4	43·0	36·26
,, 150 ,, .	1·6	9·0	10·27
Through 150 ,, .	2·8	33·5	47·09

The wear of the flints and liners is closely connected with the amount of the feed, and also the thickness of the pulp. The best results are obtained by paying close attention to these points, but no broad rules have been laid down. By repeated trials the best methods for particular ores have been arrived at in certain mills. S. Robinson states[‡] that the pulp should be very thick, containing not more than 33 per cent. of water, and Trewartha-James insists[§] on a good layer of pulp between the flints, and as a protection to the lining. Otherwise the flints grind one another and the lining instead of the ore. Alfred James points out[||] that good results have been obtained with a pulp of 50 per cent. thickness. In the Robinson Deep Mine the water was removed from the pulp as much as possible. By bringing the spitzkasten directly over the feed funnel of the tube mill a mixture of 75 parts of water with 100 of ore was obtained.[¶]

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[*] Discussion at Meeting of Inst. of M. and M., Nov. 17, 1904.

[†] *Idem*, Oct. 20, 1904.

[‡] *Idem*.

[§] *Idem*.

[||] *Idem*, Jan. 19, 1905.

[¶] W. R. Dowling, *J. Chem. Met. and Mng. Soc. of S.A.*, May, 1905, p. 304.

A section of the *Smidth-Davidsen Mill* is shown in Fig. 52, which is from a drawing kindly supplied by the Cyanide Plant Supply Company. It may be seen that the feed is through the hollow trunnion, but the discharge is through a grating near the periphery. In other mills the discharge, like the feed, is central. It is stated that this is less advantageous, resulting in a higher percentage of oversize leaving the mill uncrushed.

*The Krupp Tube Mill** differs from the Smidth-Davidsen Mill, largely in the method of charging and discharging the pulp. Fig. 53 is a longitudinal section of a mill for dry crushing, Fig. 54 a plan of the mill, and Fig. 55 a cross-section of the tube or drum, showing the discharge apertures at *c*. The drum is 1·2 metres (3 feet $11\frac{1}{4}$ inches) in diameter, and 5 metres (16 feet 5 inches) long, inside measurement, and revolves at the rate of 29 revolutions per minute. The drum consists of sheet iron 12 mm. ($\frac{15}{32}$ inch) thick, has cast-steel ends, and is lined with hard cast-iron plates. The grinding balls are introduced through a manhole, whilst the material to be crushed is fed and discharged through the hollow trunnions cast together with the end walls. The material passes first of all into the hopper, *a*, in which a studded shaft rotates. A longitudinally-grooved drum, *b*, regulates the supply of material. This drum is rotated by toothed-wheel gearing and a pair of five-step pulleys at different speeds, and contains a few balls which serve to shake the drum, and thereby secure the emptying

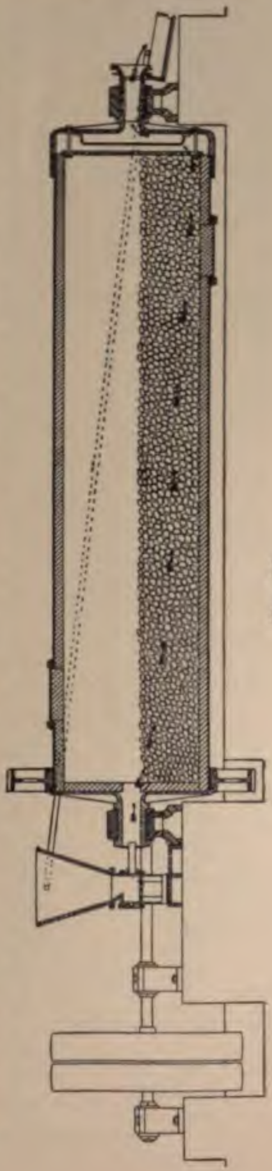

Fig. 52.

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* From description by H. Fischer in *Zeitsch. des Vereines d. Deutscher Ingénieure*, March 26, 1904.

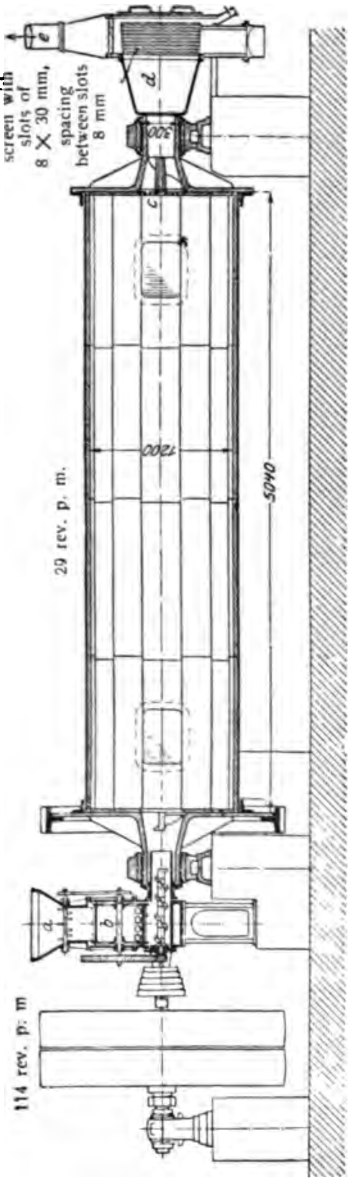
perforated screen with slots of 8 × 30 mm, spacing between slots 8 mm
e
d
300
c
1200
29 rev. p. m.
5040
a
b
114 rev. p. m

Fig. 53.

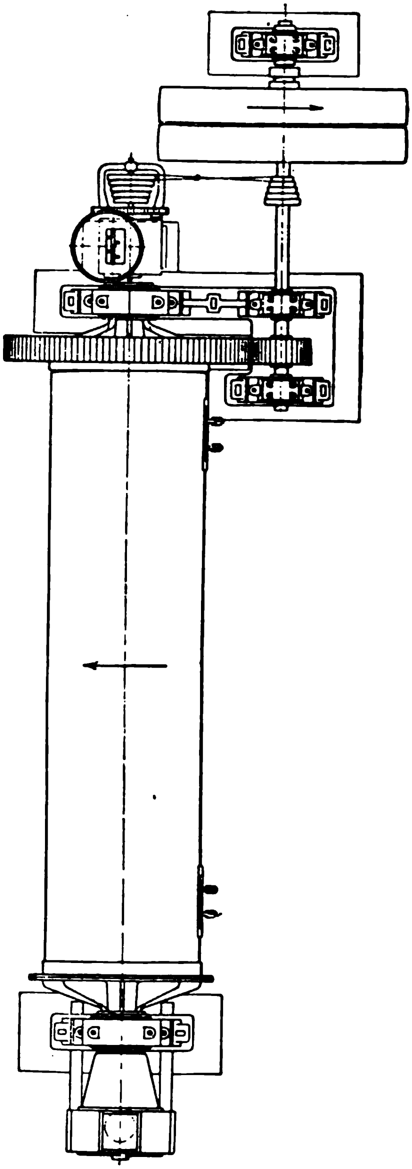

Fig. 54.

of its grooves. The material supplied is conveyed by a screw into the hollow trunnion, which is conical inside, enlarging towards the drum. Here it is scooped up by two helical blades which pass it into the drum, and prevent the balls from being thrown out of the mill. The ore then passes through the drum and through the grating, *c* (Figs. 55 and 56), into the outlet trunnion.

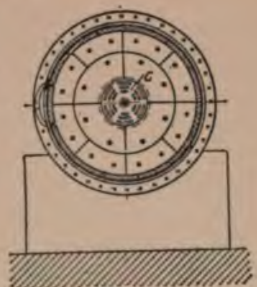

Fig. 55.

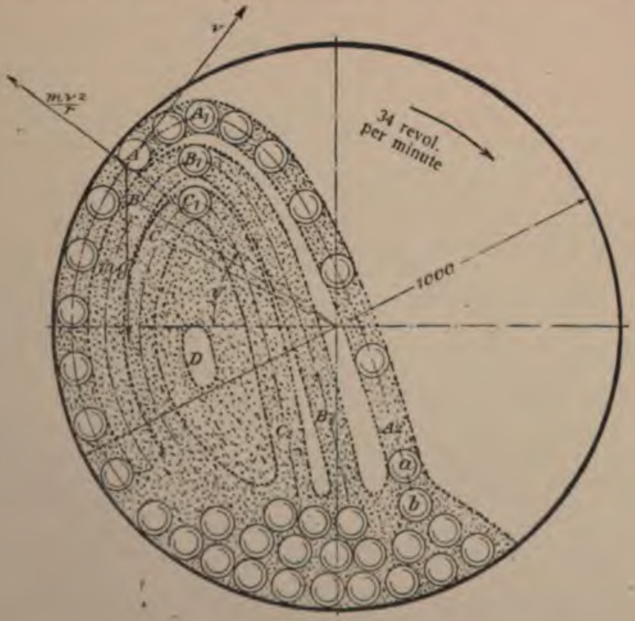

Fig. 56.

The grating, *c*, is provided with curved slots of about 25 millimetres (1 inch) in width. The material passes therefrom into a hopper, *d*, attached to the trunnion. A perforated screen is connected to the hopper. The slots of the screen are 8 millimetres ($\frac{5}{16}$ inch) in width and 30 millimetres ($1\frac{3}{16}$ inches) in length; they permit the sufficiently-crushed material to drop through, whilst very hard particles which are not crushed, and splinters of the flintstone balls, are retained and subsequently drop out on the right side (Fig. 53). The sieve is enclosed by a casing, out of which the air is drawn by means of the pipe, *e*, so that air enters the drum at all other apertures. This is to avoid loss by dusting.

Theory of Tube Mills.—According to H. Fischer,* who made experiments for the Krupp firm, the tube mill does its work mainly by impact, not by grinding. Glass drums and drums with gratings at the ends were constructed, and the action in the interior observed. One such drum 1 metre (39·3 inches) in diameter was filled with flint balls to a height of 450 mm. (17·7 inches) and rotated. Prof. Fischer found that, at the slow speed of rotation of 21 to 23 revolutions per minute, the balls rolled slowly down the slope. At 32 revolutions per minute (the correct speed for a tube of this diameter, according to the formulæ given above) the charge of balls had become looser, and their bulk was considerably more than half the capacity of the drum. At 34 revolutions (Fig. 56) the balls (A A_1, &c.) next the drum were carried up without sliding or rolling on the drum, until at a certain height they separated from it and were projected outwards in a curve, falling near the other side of the charge of pebbles. Prof. Fischer concluded that at this speed each ball fell separately, crushing and spattering the cushion of ore between it and the balls that had previously fallen. The other layers of pebbles described similar but shorter courses, and a hollow space, D, continually varying in size and shape, was always recognisable. The blow struck by each ball of the outer layer was a sliding one, the relative velocity of two balls at the moment of impact being 3·5 metres per second in the line joining their centres, and 1·2 metres per second in the direction at right angles to this. These velocities give some measure of the respective effects of the impact and grinding actions.

It is, as yet, too soon to say how far these conclusions of Prof. Fischer would apply to the conditions of actual work in practice. According to one view, in the mill with peripheral discharge, the sands pass right through the mass of the flints, which continually roll over and rub against one another, and the crushing is presumably effected by abrasion and not by impact (see Fig. 52). A. W. Warwick's view† is that, with a small charge of pebbles and a high velocity of revolution, the impact effect would predominate, and with a large charge of pebbles and a low velocity the grinding effect would be paramount. The charge of pebbles in the Smidth-Davidsen mill is 44 lbs. per cubic foot, and in the Krupp mill from 50 to 55 lbs. per cubic foot.

H. A. White agrees ‡ with Fischer that the pulverising action is due almost entirely to actual impact of the falling balls in dry crushing, but points out that where the tube is half full of water the effect of impact is of less importance. A ball falling into 2 or 3 feet of water will not strike the bottom with enough force to do much crushing, and in this case it is probable that the grinding action between the balls will do a great proportion of the work.

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* *Loc. cit.*

† *Trans. Western Assoc. of Technical Chemists and Metallurgists*, Denver Meeting, Feb. 1905.

‡ *Journ. Chem. Met. and Mng. Soc. of S.A.*, May, 1905, p. 290.

In connection with this conclusion, it is to be remembered that the best results have been obtained with very thick pulp, containing 50 per cent., or even less, of water. In a tube mill containing such pulp there would be no "free" water for the balls to fall through, and it is likely enough that the pulverising action would be almost the same as in dry crushing.

White finds that the average fall of the balls is at a maximum, if the mill is half-filled with balls, when the number of revolutions per minute, N, is equal to $\frac{34 \cdot 22}{\sqrt{D}}$, where D is the diameter of the mill expressed in metres. The formulæ of the manufacturers given above (p. 203) correspond to a slightly lower speed of revolution, but White's result, obtained by calculation, gives the same speed as that at which Fischer found by experiment that each ball fell separately.

S. H. Pearce found* at the Glen Deep Mill that the introduction of a new lining, made of rings of manganese, was attended with a cessation of the usual rumbling noise. There was a tendency for the pebbles to wear flat, and the crushing efficiency dropped. After a time the rumbling began again, and the crushing efficiency increased. He supposed that the pebbles slipped on the new smooth surface, but that the lining acquired a rough surface by wear, and the pebbles were raised higher, causing crushing [by impact] instead of reduction by grinding.

W. R. Dowling† confirms this view from the experience at the Robinson Deep Mill. He found that a silex lining takes a larger feed and gives a finer ground product than manganese steel, owing to the absence of sliding action when the former is used.

Tube Mill Results in the Transvaal.—The results of the first tube mill runs at the Treasury Mine in the Transvaal in 1904-5 have been given by H. Leupold.‡ A Krupp mill, 3 feet 6 inches in diameter and 13 feet in length, inside measure, was used, the shell liners being of cast iron 1 inch thick. The charge consisted of 7,000 lbs. of pebbles of from 2 to 3 inches in diameter. The best speed was about 33 revolutions per minute, and the power used was computed to be only 20 H.P. under favourable conditions, which seems rather low. Mercury was apparently charged in with the pulp, which consisted of concentrates from a 60-stamp battery, crushed through a screen with 700 holes to the square inch during the first run, and a screen with 300 holes during the greater part of the second run. About 2,200 lbs. of concentrates were passed through the mill per hour. The results were as follows:—

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* *Idem*, p. 304. † *Idem*.

‡ *Trans. S.A. Assoc. of Eng.*, April, 1905. *Page's Weekly*, June 2, 1905, p. 1198.

	First Run.	Second Run.
Running time, . . .	955 hrs. 45 mins.	1,123 hrs. 55 mins.
Weight of concentrates treated,	1,050 tons	1,240 tons
,, liners consumed, .	3,115 lbs.	3,702 lbs.
,, pebbles ,, .	4,191 ,,	4,280 ,,
Mercury lost, ...	85 ozs.	70 ozs.
Amalgam won, . . .	575 ,,	718 ,,
Fine gold ,, . . .	168 ,,	255 ,,
Working costs, . . .	3s. 8½d. per ton of concentrates, or 4·35d. per ton of ore milled.	3s. per ton of concentrates, or 3·29d. per ton of ore milled.

By the change in the size of the screen holes, the stamp duty was increased from 4·75 to 5·20 tons per 24 hours. In each run new lining plates were worn down to about ⅜ inch thick. They can be used until worn to a thickness of ⅛ inch. It was found that the end liners lasted about twice as long as the circumferential plates. The ground concentrates contained 85 per cent. of material which would pass through an 80-mesh sieve (6,400 holes to the square inch). They were cyanided with the bulk of the sands and slimes, and no increase in the consumption of KCy per ton was observed, though it had been expected, owing to the iron ground off the liners.

At the Robinson Deep, work with a tube mill was begun in April, 1905.* The mill was 22 feet long and 4 feet 9 inches in internal diameter, and made 29 revolutions per minute. The pulp from the stamps, which crush through "light 1,000-mesh" screens, is de-watered in a spitzkasten 3 feet square, and after passing through the tube mill is again mixed with the water, and led over four shaking plates with 190 oscillations per minute. Tests showed that a feed of 300 tons per day gave a higher grinding efficiency than one of 165 tons per day, although the product in the former case was only partially ground. Silex liners did more grinding work than steel liners. The tube mill was found to be the best trap for escaped amalgam from the battery, and a very rich product was found in cleaning the mill out to renew its liners.

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* W. L. Dowling, *Journ. S.A. Assoc. of Engineers*, May, 1905. *J. Chem. Met. and M. Soc. of S.A.*, August, 1905.

CHAPTER XII.

ROASTING.

Introduction.—The operation of roasting, as a preliminary to chlorination, has for its object the expulsion of the sulphur, arsenic, antimony and other volatile substances existing in the ore, and the oxidation of the metals left behind, so as to leave nothing (except metallic gold) which can combine with chlorine when the ore is subsequently treated with it in aqueous solution. For this purpose the ore is heated in a furnace, through which a current of air is passed, salt being added if oxide of copper, lime, magnesia, &c., are present. Ores containing much pyrites might be freed from most of their sulphur by pile roasting, and then subjected to fine crushing and a dead roast in a reverberatory furnace, but the extra cost of handling would probably exceed the saving due to the smaller consumption of fuel. This system has not been tried in chlorination mills on an extensive scale. The ordinary reverberatory furnace, worked by hand labour, is still in use, especially where only a few tons, or less, of concentrates are to be treated per day. Various mechanical furnaces, capable of handling large quantities of ore, have been devised to supersede the old-fashioned contrivance, and some of these will be described in the sequel.

Hand-worked Reverberatory Furnace.—The construction of the ordinary reverberatory furnace is too well known to need detailed description here.* It consists of a vaulted chamber, containing the ore; through this chamber, the flames and products of combustion from a furnace and a current of air are made to pass in a horizontal direction above the ore, which is thus heated. The ore is also stirred by hand with iron rakes, which are passed through small working doors. The hearth of the vault (also called the "laboratory" of the furnace) is formed of bricks placed on edge, as close together as possible. No mortar is used, but a little clay is plastered between the bricks. The height of the furnace hearth is about 3½ feet above the floor of the building, on which the labourers stand, and the space underneath the hearth is either occupied by vaults or filled with well tamped rubble. The arch is usually one course of bricks (8 inches) thick; the height between it and the hearth is, in long furnaces, about 24 inches near the bridge, and gradually diminishes towards the other end. This height is less in short furnaces. The best firebricks are used for the fire-box and bridge, and for the hearth and arch of the first few feet of the "laboratory." The remainder is made of common brick. It is necessary to have a damper in the flue to regulate the draught; the aperture of the flue should not be on a level with the hearth, as in that case the loss by dusting is increased. The brickwork of the furnace is supported by longitudinal and transverse iron braces. The working doors have cast-iron frames, and are about 15 inches wide and 9 inches high. The fuel used must of course be a long-flame coal, or wood; short-flame coal and coke are inadmissible.

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* See the *Introduction to the Study of Metallurgy*, by Prof. Roberts-Austen, 1902, p. 305. Also Küstel's *Roasting of Gold and Silver Ores*, 1880, pp. 80-89; and Grüner's *Traité de Metallurgie*, p. 265.

For the particular purpose of roasting pyritic ores before chlorination, the temperature on the working floor of the furnace is kept low when the ore is first charged in, and high in the later stages. If a single small floor is used the fire must be alternately checked and urged to secure these conditions. Moreover, when the roasting is nearly complete, the high temperature required renders the gases passing into the flue very hot, and so a corresponding waste of fuel results. To prevent this waste, it is customary for roasting furnaces to be built with a very long hearth, or to have several successive hearths, so as to utilise the waste heat from the portion of the working floor next the fire. Furnaces with three floors at slightly different levels are much favoured; in these a charge remains for a few hours on each of the floors in succession. It is first placed on the floor farthest removed from the fire, and, after a time, is raked down on to the middle hearth, and thence to that nearest the fire, fresh charges being put on the spaces just cleared, so that there are always three charges in the furnace in various stages of oxidation.

The most usual form in the Western States of America is the "4-hearth," in which the length of the hearth is four times its width, so that the dimensions are, say, width 15 feet, length 60 feet. In this case there should be eight working doors on each side. Instead of three floors at different levels, a single continuous floor, gently sloping from the flue towards the fire, is in use at many works in Australia, Mexico, and the United States. At Suter Creek a continuous-hearth furnace was 12 feet wide and 80 feet long, and the mineral was worked in three distinct parts, as though there were three floors. The angle of slope is made large in some Australian furnaces, so that in the course of the "rabbling" or stirring, the ore continually travels towards the fire-box. Furnaces with two or three superposed floors are also used to a limited extent; the lowest floor is next the fire-box, and communicates by a vertical flue with the floor above, and so on. The ore is charged-in on the top floor, and after a time is raked down through the vertical flue on to the next floor. In this case the floors are heated by the gases passing below them as well as above them, and fuel is economised, but the furnaces are costly to build and to keep in repair. Merton's furnace, subsequently described, is one of this kind with automatic rabbling.

It was proposed many years ago to insert a drop of 10 feet between the finishing floor and the floor next to it. The charge, when already red hot, would thus fall vertically downwards in a thin shower against a current of hot air. Some furnaces are said to have been built in this way, but it seems that none are now in existence. The principle is excellent, and is utilised in the Stetefeldt furnace.

When the pyrites to be roasted are rich, it may be an advantage to build dust chambers to ordinary reverberatory furnaces. The amount of gold contained in the dust thus recovered is usually only 1 or 2 per cent. of that contained in the ore, so that in some cases it may be a long while before the dust chamber pays for itself, even if that point is ever reached.

The operation of roasting pyrites in an ordinary furnace with three floors may be described as follows :—The furnace being hot, and the flame from the fire-box reaching completely across the first flour, the ore is charged-in on the third floor and spread out by the rabbling tool. The weight of the charge may be taken as from 12 to 18 pounds per square foot of floor space, varying according to the nature of the ore, a high percentage of sulphur necessitating small charges. The layer of ore is 2 or 3 inches deep. It is not spread quite evenly, but made to form a series of parallel ridges by means of the rabbling tool, so as to increase the surface exposed to the air. The working doors may be closed at first to heat the ore quickly. Moisture is at once given off in great quantities, and the sulphur soon begins to burn with a blue flame. When this is seen to take place, all the working doors are opened, and the charge is energetically rabbled, with little intermission, until the sulphur flame disappears. If this is not done, clots are formed which are afterwards difficult to break up. The air for the combustion of the sulphur is supplied by holes in the fire-bridge and from the working doors. The flames and heated products of combustion from the fire tend to rise above the colder air, and move along next to the arch of the furnace, while the air forms a sheet between these gases and the ore. In practice, although the air is introduced below them, nevertheless the reducing gases from the fire partially mix with the air, and greatly reduce its oxidising power. Moreover, the combustion of the sulphur in the ore on the first two floors further reduces the amount of free oxygen present in the current of air, and roasting on the third floor is, therefore, largely dependent on air derived from the working doors.

When the sulphur flames have abated, and the charge has been heated almost to redness, it is transferred to the middle floor, where it is raised to a dull red heat and most of the sulphating takes place. During this stage the ore swells considerably, so as to occupy much more than its original bulk. All the lumps previously formed should be broken up on this floor. Rabbling is continued until the ore is uniformly dull throughout, so that, on turning it over, the fresh surfaces appear but little brighter than that which has been exposed for some time. The charge is then transferred to the floor next the fire. There is now little risk of the formation of lumps, and the charge may be allowed to reach a bright red heat. Rabbling is of less importance than before, as little oxidation takes place, the chief reaction which occurs being the decomposition of the sulphates already formed. As long as this is still going on, the ore emits the odour of sulphur dioxide. When no further odour can be detected, and the ore can be piled up so as to maintain a vertical face, shows no bright specks on its glowing surface, emits no sparks if some of it is tossed up by the working tool, and is inclined to become black very readily from cooling, the charge is said to be "dead" or "sweet," and is ready to be withdrawn. It should be observed that, when the ore contains much sulphur, its particles at a low red heat appear less coherent than when cold, and flow almost like water so that the charge cannot be made to form a heap with steep sides. Care must therefore be taken when the ore is on the middle floor to prevent any part of the charge from flowing out of the working doors, which it is very liable to do when being rabbled.

Küstel states* that the best means of rapidly ascertaining whether a charge is completely roasted is to throw a little of the ore into some water, and then to plunge a bright iron rod into the liquid. If the rod remains bright the ore is ready for withdrawal, but if sulphates still remain undecomposed, the surface of the iron will be darkened. This is not a safe test with all classes of ore, as the presence of sulphate of iron in the water would not be detected in this way. A more trustworthy and equally simple test is to add a few drops of chloride of barium to the water. A white cloud, which consists of $BaSO_4$, indicates the presence of soluble sulphates. In most cases the water need not be filtered before it is tested, but even when filtration is necessary the whole operation can be performed in two or three minutes.

Certain insoluble sulphides may be detected by boiling the roasted ore with caustic potash, filtering, and adding a solution of lead carbonate. A brown colour or black precipitate denotes the presence of sulphides. This test is used at the Great Boulder, according to D. Clark.*

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* *Roasting of Gold and Silver Ores.* San Francisco, 1880.

Although roasting is carried on until the ore is said to be "dead," it is not practicable to eliminate the whole of the sulphur, and a small percentage both of insoluble and of soluble sulphur can be found in roasted ores. The insoluble sulphur is, of course, more detrimental to gold extraction by chlorine than is soluble sulphur. It is not usual in roasting for chlorination to leave more than about 0·1 to 0·15 per cent. of insoluble sulphur in the ore, but much larger percentages of soluble sulphates are often left undecomposed in roasted ore.

The charge is withdrawn by a scraper, and falls by gravity through a hole in the floor of the furnace near the working door into a pit below. This hole is covered by a plate while roasting is being performed.

The time of roasting depends chiefly on the ore, but may be shortened by more continuous rabbling than most workmen can perform, the work being somewhat exhausting. In a three-floor furnace, concentrates with 15 per cent. of sulphur usually remain eight hours on each of the three floors. The fuel used is either wood or flaming-coal. If the flame from the coal is not long enough to reach across the first floor, this will not be heated uniformly; in that case the part of the charge next the fire-bridge is finished first and must be moved away, while that from the other end of the floor is brought up nearer to the fire. This causes a great increase in the labour, besides occasionally leading to the withdrawal of a charge of which a part is not quite "dead." The draught is regulated by the damper in the flue leading to the dust chamber, and by opening and closing the working doors.

The consumption of coal for fuel is usually from 10 to 20 per cent. of the weight of the ore.

Chemistry of Oxidising Roasting.—Sir Wm. Roberts-Austen discusses as follows † the roasting of a "mixture consisting of sulphides mainly of iron and copper, with some sulphide of lead, small quantities of arsenic and antimony as arsenides, antimonides, and sulpho-salts, usually with copper as a base. The temperature of the furnace in which the operation is to be performed is gradually raised, the atmosphere being an oxidising one. The first effect of the elevation of the temperature is to distil off sulphur, reducing the sulphides to a lower stage of sulphurisation. This sulphur burns in the furnace atmosphere to sulphurous anhydride (SO_2), and coming in contact with the material undergoing oxidation is converted into sulphuric anhydride (SO_3). It should be noted that the material of the brickwork does not intervene in the reactions, except by its presence as a hot porous mass, but its influence is, nevertheless, considerable. The roasting of these sulphides presents a good case for the study of chemical equilibrium. As soon as the sulphurous anhydride reaches a certain tension, the oxidation of the sulphide is arrested, even though an excess of oxygen be present, and the oxidation is not resumed until the actions of the draught change the conditions of the atmosphere of the furnace, when the lower sulphides remaining are slowly oxidised, the copper sulphide being converted into copper sulphate, mainly by the intervention of the sulphuric anhydride, formed as indicated. Probably by far the greater part of the iron sulphide only becomes sulphate for a very brief period, being decomposed into the oxides of iron, mainly ferric oxide, the sulphur passing off. Any silver sulphide that is present would have been converted into metallic silver at the outset were it not for the simultaneous presence of other sulphides, notably those of copper and of iron, which enables the silver sulphide to become converted into sulphate. The lead sulphide is also converted into sulphate at this low temperature (viz., about 500°). The heat is now raised still further with a view to split up the sulphate of copper, the decomposition of which leaves oxide of copper. If, as in this case, the bases are weak, the sulphuric anhydride escapes mainly as such; but when the sulphates of stronger bases are decomposed the sulphuric anhydride is to a great extent decomposed into a mixture of sulphurous anhydride and oxygen. The sulphuric anhydride, resulting from the decomposition of this copper sulphate, converts the silver into sulphate, and maintains it as such, just as, in turn, at a lower temperature, the copper itself had been maintained in the form of sulphate by the sulphuric anhydride eliminated from the iron sulphide. When only a little of the copper sulphate remains undecomposed, the silver sulphate begins to split up (viz., at about 700°) . . . partly by the direct action of heat alone, and partly by reactions such as those shown in the following equations:—

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* *Australian Mining and Met.*, p. 24.

† Presidential address to the Chemical Section, British Association, Cardiff Meeting, 1891.

$$\begin{aligned} Ag_2SO_4 + 4Fe_3O_4 &= 2Ag + 6Fe_2O_3 + SO_2 \\ Ag_2SO_4 + Cu_2O &= 2Ag + CuSO_4 + CuO \end{aligned}$$

The charge still contains lead sulphate, which cannot be completely decomposed at any temperature attainable in the roasting furnace except in the presence of silica. . . . The elimination of arsenic and antimony gives rise to problems of much interest, and again confronts the smelter with a case of chemical equilibrium. For the sake of brevity it will be well for the present to limit the consideration to the removal of antimony, which may be supposed to be present as sulphide. Some sulphide of antimony is distilled off, but this is not its only mode of escape. An attempt to remove antimony by rapid oxidation would be attended with the danger of converting it into insoluble antimoniates of the metals present in the charge. In the early stages of the roasting it is, therefore, necessary to employ a very low temperature, and the presence of steam is found to be useful as a source of hydrogen, which removes sulphur as hydrogen sulphide, the gas being freely evolved. The reaction

$$Sb_2S_3 + 3H_2 = 3H_2S + 2Sb$$

between hydrogen and sulphide of antimony is, however, endothermic, and could not, therefore, take place without the aid which is afforded by external heat. The facts appear to be as follows :—Sulphide of antimony, when heated, dissociates, and the tension of the sulphur vapour would produce a state of equilibrium if the sulphur thus liberated were not seized by the hydrogen, and removed from the system. The equilibrium is thus destroyed, and fresh sulphide is dissociated. The general result being that the equilibrium is continually restored and destroyed until the sulphide is decomposed. The antimony combines with oxygen and escapes as volatile oxide, as does also the arsenic, a portion of which is volatilised as sulphide.

"The main object of the process which has been considered is the formation of soluble sulphate of silver." The reactions, however, are precisely similar in an ordinary oxidising roast.

The following remarks on the decomposition of the various minerals present in complex ores may be of use in assisting the student to understand the reactions which proceed in the roasting furnace :—

1. *Iron Pyrites*, FeS_2.—On heating this compound sulphur is volatilised, the reactions being probably expressed thus :—

$$3FeS_2 = Fe_3S_4 + S_2$$
$$7FeS_2 = Fe_7S_8 + 3S_2$$

The sulphur burns to SO_2, which is partly converted by the heated quartz, &c.,* into SO_3, uniting with the free oxygen present. The ferrous sulphate formed by this sulphuric acid is split up by the heat and the ferrous oxide (FeO) converted into ferric oxide (Fe_2O_3) which gives the ore a red colour when cold. Some basic sulphates always remain undecomposed. If the temperature of the part of the charge next the fire-bridge has been too high, or if the charge is kept too long in the furnace, especially when not freely exposed to the air, some magnetic oxide is formed, thus :—

$$3Fe_2O_3 = 2Fe_3O_4 + O$$

The presence of magnetic oxide makes the ore darker in colour.

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* Plattner's *Metallurgische Röstprozesse*, Freiberg, 1856.

This is an undesirable change, as the magnetic oxide is acted on by chlorine far more readily than the sesquioxide.

Donald Clark states * that magnetic pyrites, Fe_3S_4, produced at an early stage is oxidised direct to Fe_3O_4 if the temperature is moderately high and the supply of air ample. When this is brought into the hotter part of the furnace, the magnetic oxide is slowly converted to ferric sesquioxide, Fe_2O_3. He gives no evidence in support of these statements, which can only be true of a small portion of the charge, if any.

W. E. Greenawalt † states that the dark magnetic oxide can be reconverted to the red sesquioxide by subjection to a lower temperature and an abundant supply of air. For this reason he advocates finishing at a lower temperature than is used for the previous stages of roasting. In four successive tests on 100-ton lots he found that a higher percentage of extraction by chlorination was obtainable by roasting at a high initial heat and a low finishing heat than by roasting at a lower initial heat and a higher finishing heat, although, in the latter case, the elimination of the sulphur was more complete.

2. *Copper Pyrites.*—The decomposition of the copper sulphate formed in the furnace leaves a mixture of cuprous and cupric oxides, both soluble in chlorine.

3. *Galena*, PbS.—The presence of this mineral in any but small quantities is very detrimental, as both lead sulphate and lead silicate (formed by its decomposition in the presence of silica) are very fusible, and, at the temperature required to split up copper sulphate, cause the ore to become pasty and form lumps. Roasting must be performed very slowly and cautiously to avoid this effect.

4. *Arsenical Pyrites*, FeAsS.—Arseniates of iron, copper, lead, &c., when formed are not easily decomposed, as they resist a high temperature, and are only slowly converted into sulphates by sulphuric acid at a red heat. It is, therefore, desirable to avoid their formation, and with this end in view the precautions which have been already mentioned above are taken.

5. *Antimonial Sulphides* are still more difficult to deal with, the antimoniates formed being less easily decomposed than arseniates. Their formation is avoided in the manner already described.

6. *Blende*, ZnS, forms oxide and sulphate of zinc, of which the latter can only be split up by a very high temperature. At a bright red heat a basic sulphate is formed which is converted to oxide at a white heat. If blende is roasted at a high temperature and with a plentiful supply of air, sulphate of zinc is not formed to a large extent.

7. *Carbonate of Lime*, $CaCO_3$, is decomposed at a red heat,

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* *Australian Mining and Metallurgy*, p. 289.

† *Eng. and Mng. Journ.*, July 29, 1905, p. 145.

CO_2 being given off and caustic lime, CaO, left in the charge. The change is slow at 600° (low red heat) and rapid at 800° (full red heat). Magnesium carbonate is similarly decomposed.

8. *Tellurides* containing gold are fusible far below a red heat, and heavy losses of gold may occur through absorption by the furnace bottom. The melted tellurides of gold may remain in the ore, and in that case at a red heat the tellurium is partly volatilised and partly oxidised to the oxide TeO_2, which also sublimes. Spherical beads of gold remain behind and are difficult to dissolve. *Selenides* behave similarly.

9. Metallic *Gold*, *Silver*, &c., are fused at high temperatures, forming spherical beads which are difficult to dissolve. The temperature of the ore should never be allowed to exceed 1,000°, for this reason.

Elimination of Arsenic and Antimony.—H. M. Howe, in explaining how this is effected, distinguishes three horizontal zones in the ore:* (1) the upper surface, where oxidation is only slightly hindered by sulphurous and sulphuric acids and by the products of combustion of the fuel; (2) the middle layers where oxidation proceeds to a very limited extent; (3) the lowest layers, where "a pellet of ore is simply exposed to the action of the other pellets with which it is in contact, of volatilised sulphur, and of sulphurous and sulphuric anhydrides generated by the action of sulphur on previously formed metallic oxides." He proceeds—"The expulsion of arsenic and antimony as sulphides is favoured in the middle and lower zones by the presence of volatilised sulphur, mixed with sulphurous acid and at most a very limited supply of free oxygen and sulphuric acid. In the upper part of the middle layer, to which a small amount of free oxygen penetrates, we have the gently oxidising conditions favourable to the formation of arsenious acid and trioxide of antimony. In the upper zone the stronger oxidising conditions rather favour the formation of fixed arseniates and antimoniates, though, even here, part of the arsenic and antimony may volatilise and escape while passing through their intermediate volatile condition of arsenious acid and trioxide of antimony." On stirring the mass, these arseniates and antimoniates, being exposed to the reducing action of volatilised sulphur and undecomposed sulphides in the lower zones, may again be converted into volatile oxides. Protoxide of iron, suboxide of copper, and sulphurous acid are also efficacious in reducing arsenic acid, higher oxides of iron and copper, and sulphuric acid being formed. "Thus, every individual atom of arsenic may travel forth and back many times through the volatile condition, being oxidised at the surface and reduced below the surface, . . . and every time it arrives at this volatile condition an opportunity is offered it to volatilise and escape." If a small quantity of coal or coke dust is mixed with the ore, after it has been completely oxidised, and the air excluded, the arseniates and antimoniates are again reduced to the lower oxides, and, if they are "carried past the volatile state," *i.e.*, reduced to metals, they may be again passed through it by an oxidising atmosphere. "Of course the expulsion of arsenic and antimony is favoured by the presence of a large proportion of pyrites, both because the sulphur distilled from the pyrites tends to drag them off as sulphides, and because the presence of the pyrites prolongs the roasting, and thus increases the number of times which the arsenic and antimony pass back and forth past their volatile conditions; hence, it is sometimes desirable to mix pyrites with impure ores to further the expulsion of their impurities."

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* Copper Smelting, *Bulletin* No. 26 *U.S.A. Geol. Survey*, Washington, 1885.

The Use of Salt in Roasting.—Certain ores require the addition of salt in roasting in order to chloridise material which would otherwise absorb chlorine when the ore came to be "gassed," and so cause additional expense as well as inconvenience. If silver as well as gold is to be extracted from the ore, the addition of salt is necessary in order to form chloride of silver in the furnace, since metallic silver is not attacked by chlorine at the highest temperature ever employed in the leaching vat. The silver chloride is then dissolved out by hyposulphite of soda or some other solvent either before or after the extraction of the gold.

Even if no silver is present, an ore must be roasted with salt if it contains much copper (as sulphide, or as an oxidised salt), lime, magnesia, or other substance which, after being subjected to an oxidising roasting, is rapidly attacked by chlorine at ordinary temperatures. The salt is usually added towards the end of the operation, when no sulphides and only a small percentage of sulphates are left undecomposed; sometimes, however, the ore and salt are mixed before charging-in. To some sulphides only 5 pounds of salt per ton of ore are added, but others require as much as 90 pounds per ton. The weight of salt added must be at least six to eight times that of the silver present in the ore. If a large amount of salt is used, it is desirable to leach the roasted ore with water, before treating it with chlorine gas, in order to remove the coating of soluble sulphates and chlorides remaining on the surface of the granules of ore.

The chemical action of the salt is due to a double decomposition between it and the sulphates of the heavy metals, by which sulphate of soda and the chlorides of the heavy metals are produced. The following general equation approximately represents the reaction:—

$$2NaCl + RSO_4 = RCl_2 + Na_2SO_4$$

Chlorine is also set free by the action of sulphuric anhydride on salt, and the presence of water vapour induces the formation of much hydrochloric acid. These gases act directly on the several constituents in the ore, forming chlorides and oxychlorides. The metallic chlorides and oxychlorides formed are in many cases volatile (*e.g.*, the compounds of copper, iron, lead, arsenic, antimony, &c.), and, in passing off, the volatile compounds carry away with them varying proportions of gold and silver, which, as a rule, are not recoverable in the dust chambers. The chloride of copper is especially active in causing these losses.

Other reactions which probably take place are as follows :—

1. Ferrous sulphate, acted on by salt at a red heat in presence of air, yields hydrochloric acid and chlorine, which act on the gold and silver, while ferric sesquioxide and sodic sulphate are produced.

2. Ferric chloride, Fe_2Cl_6, is also produced at the same time. This is volatile, and chloridises silver with great energy at a red heat, sesquioxide of iron being produced.

3. Cupric chloride, $CuCl_2$, is easily decomposed into cuprous chloride, Cu_2Cl_2, and free chlorine, or into the oxychloride, $Cu_2O . Cl_2$, and free chlorine. The vapours of $CuCl_2$ thus give rise to further supplies of nascent chlorine available for the chlorination of the silver.

4. Arsenic and antimony form volatile chlorides which are decomposed by means of oxygen and water vapour, yielding arsenious and antimonious acids and nascent chlorine or hydrochloric acid.

It is thus obvious that the presence of base minerals is advantageous in that they may cause nascent chlorine to be set free in the presence of silver in all parts of the furnace. On the other hand, the loss of gold is increased by any increase in the quantities either of silver or of the base metals, since in the former case the time of the roasting is prolonged. The best chloridising effect is obtained in a highly oxidising atmosphere, so that very little sulphur is required in the ore, and, if much is present, the practice of eliminating the greater part before adding the salt is not likely to be attended with any diminution in the percentage of silver chloride formed. Moreover, the water vapour in the furnace gases promotes the formation of hydrochloric acid. When salt is used in roasting, the ore is often allowed to cool slowly in heaps after being withdrawn from the furnace. If treated in this way, a higher percentage of the silver, &c., is found to be chloridised than if the ore is wetted down at once, or even spread out to cool in a thin layer. On the other hand, the losses of gold by volatilisation are increased by slow cooling. Chlorine continues to be evolved for a long time after the withdrawal of the charge has taken place, the heaps smelling strongly of the gas.

Losses of Gold in Roasting.—Plattner proved in 1856* that in the oxidising roasting of ordinary auriferous pyrites, a loss of gold can take place only when the operation is carried on so rapidly that fine particles are carried off mechanically by the draught. This conclusion, as far as sulphides and arsenides are concerned, has been confirmed by Kustel,† and by Prof. S. B. Christy,‡ but the latter adds that it is extremely difficult to prevent all mechanical loss by dusting, which is caused by even a moderate draught. Kustel records the loss of 20 per cent. of the gold present during the oxidising roasting of certain tellurides of gold and silver, and states that this is not a mechanical loss, but is due to volatilisation. This seems to be a mistake, see p. 7.

The losses of gold which are sustained when salt is added to the furnace charge may be very great. Kustel found that a telluride ore, on being roasted with 4 per cent. of salt, lost 8 per cent. of its gold before the ore was red hot. Aaron§ found that certain ores, consisting of simple pyrites, suffered great loss of gold in roasting with salt which had been added at the commencement of the operation; only a small part of this gold was condensed in the flue, in which was found a yellowish fluffy precipitate, consisting largely of chlorides of copper and iron, and containing nearly 30 ozs. of gold to the ton. He found that the loss was greatly reduced by diminishing the quantity of salt, and by reserving it until the dead roasting was nearly complete.

In the chloridising roasting of a Mexican ore, consisting mainly of magnetite and pyrites with 3·5 to 7 per cent. of chalcopyrite, C. A. Stetefeldt found the losses of gold to be from 42·8 to 93 per cent. of the total gold contained. He states‖ that "there is no doubt that the volatilisation of the gold takes place with that of the copper chlorides. The loss increased with the quantity of these chlorides formed and volatilised." He further shows, however, that the presence of copper chloride is not the only possible cause of loss, since an ore consisting of hard white quartz, intimately mixed with about 7 per cent. of calcite and a little pyrites, lost 70 to 80 per cent. of its silver, and 68 to 85 per cent. of its gold, when roasted with 5 per cent. of salt. When subjected to an oxidising roast, no loss of gold took place. The reason for the extraordinary behaviour of this ore was not discovered.

Prof. Christy found¶ that, in the ores on which he experimented on a small scale in a muffle furnace, a greater loss was sustained by adding the salt near the end of the roasting operation than by mixing the same weight of salt with the ore at the start. He explained that this is due to the fact that the amount of gold volatilised varies with the amount of chlorine which comes in contact with it. When the salt is added at the start, the chlorine is at first removed by the sulphur as fast as it is formed, escaping as chloride of sulphur, and thus the gold is protected from attack. When the salt is added after a long oxidising roast, the chlorine is rapidly generated (the ore being red hot and containing large quantities of sulphates), and the gold is no longer protected from attack by the sulphur. The loss of gold is also in all cases increased by working at a higher temperature, owing to the large amount of chlorine generated, and to the increase in the volatility of the gold. It is apparent from the results given on p. 23 that the temperature used in chloridising roasting must be very carefully regulated, the loss of gold being increased far more by high temperature than by a lengthening of the time in the furnace. Moreover, the salt must be reduced to the least possible quantity. It must, however, be remembered that the maximum volatility of gold chloride is at about 250°, or far below a red heat.

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* *Metallurgische Röstprozesse*, Freiberg, p. 128.

† *Roasting of Gold and Silver Ores*, 1880, p. 56.

‡ *Trans. Am. Inst. Mng. Eng.*, 1888.

§ *Leaching of Gold and Silver Ores*, 1881, p. 121.

‖ *Trans. Am. Inst. Mng. Eng.*, vol. xiv., p. 339.

¶ *Trans. Am. Inst. of Mng. Eng.*, 1888.

The advantage found to be gained in practice by adding salt near the end of the operation is due to the fact that, in the continuous roasting of ores in long-bedded furnaces, the gases given off from the finishing floor pass over a great length of comparatively cold, unsalted, and unoxidised ore before reaching the flue. The quantity of gold chloride mixed with the chlorine which is evolved from the red-hot ore as soon as the salt is added is no doubt large, but the SO_2 from the colder ore, and the steam from the fuel, "offer excellent means for the reduction of the chloride of gold right within the furnace, while the most efficient means probably is the pyrites themselves," which have been proved to be readily capable of condensing gold on their surface. If all the salt is added at the start, there is a continued volatilisation of chloride of gold throughout the furnace, and a less favourable opportunity for it to condense. The difference between the results in the muffle and in the reverberatory furnace is thus explained.

At Nevada City, at the Merrifield Mine, and in other works in the neighbourhood, the old-fashioned long furnace, with a single step separating the finishing hearth from the rest of the furnace, was still used in 1888.* These furnaces are from 55 to 65 feet long, holding from 6 to 9 tons, and producing about 3 tons of roasted ore per day, so that the ore remains in the furnace from two to three days. The custom there was to give the ore a long oxidising roast at a low red heat, ending in a low cherry-red heat, and then, when the ore reached the finishing floor, the temperature was slightly lowered, and the salt added. The salt was stirred thoroughly into the ore, and as soon as it was "dissolved" by the roasted ore—*i.e.*, in about half an hour—the charge was drawn into the cooling pit. This lowering of the temperature is evidently of great importance in reducing the loss, while the duration of the roasting is regarded as less material, so long as no salt is present. These mills were on custom work, charging $15 to $17 per ton of ore for treatment, and guaranteeing a yield of 90 per cent. of the gold and 60 per cent. of the silver. Their method of roasting seems to be considered in California as that best suited to concentrates containing a high percentage of sulphur, but their loss in roasting has not been ascertained. The best method of roasting any particular ore, however, cannot be determined by any general rule, and exhaustive experiments must be made in every case before a definite course of procedure is finally adopted.

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* *Trans. Am. Inst. Mng. Eng.*, vol. xiv., p. 340.

At one of the Californian chlorination mills it was found by experiment in 1882 that nearly 50 per cent. of the gold and 28 per cent. of the silver were being lost by volatilisation. In this case the pyrites was roasted on two hearths for thirty-six hours, 1 per cent. of salt being added four hours before the charge was drawn. The reason for the great loss was thought by Professor Christy to be the high temperature of roasting, particularly on the charging-in floor.

The variation of the loss in different ores which are treated precisely alike is doubtless due partly to the presence or absence of metals forming volatile chlorides which carry off the gold, and partly to the physical condition of the latter, the volatilisation being greater if it is in a state of minute subdivision.

Croasdale found,* from experiments on a number of ores, that from 80 to 99 per cent. of the gold contained in them could be volatilised by roasting with salt.

MECHANICAL FURNACES.

The furnaces which have been designed with the object of saving the labour necessary to work the reverberatory furnaces may be divided into four classes, viz. :—

1. Stationary hearth furnaces, supplied with iron hoes moved by machinery by which the ore is rabbled. The O'Hara, Spence, Pearce Turret, Brown, Ropp, Edwards, Merton, and other furnaces are examples of this class.

2. Rotating-bed furnaces, in which the hoes or stirrers are stationary, while the bed supporting the ore revolves. The Godfrey calciner† and the old Bunker Hill furnace are examples of this class.

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* *Eng. and Mng. Journ.*, Aug. 29, 1903, p. 312.

† *Trans. Inst. Mng. and Met.*, vol. vii. (1899), p. 323.

3. Rotating cylindrical furnaces, which consist of brick-lined iron cylinders capable of being rotated, so that the ore is tumbled over and over by their motion while it is being roasted. Examples are the Brückner, the White-Howell, the Hofmann, and the Argall furnaces.

4. Shaft furnaces, in which the powdered ore falls by gravity, in a shower, through an ascending column of hot air, the oxidation being effected in the course of the fall. The Stetefeldt furnace, which is the only one based on this principle, is not used for dead roasting, as it is not adapted for the purpose. It is used for the chloridising roasting of silver ores, and will not be described in this volume.

Furnaces with Mechanical Stirrers—*O'Hara Furnace.*—This is the oldest mechanical furnace, and it bears a great resemblance to the old-fashioned reverberatory furnace. It has two superposed hearths, in each of which the arch is very low, so as to confine the heat close to the ore. An endless chain, set in motion by suitable machinery, passes through the furnace, resting on the upper hearth, and returns along the lower hearth. Attached to the chain at proper intervals are iron frames of a triangular shape; on these frames are a number of ploughs or hoes set at an angle, so that one set of hoes turns the ore to the centre, and the next set turns it in an opposite direction towards the walls. The ploughs thus stir the ore thoroughly, and at the same time move it gradually towards the fire. The ore falls from the upper to the lower hearth by gravity, and similarly falls from the lower hearth into a pit when it arrives at the hottest place in the furnace. The ore is from five to ten hours in the furnace, according to the amount of sulphur contained in it. In the modern form the hearths are each 8 feet wide and 90 feet long, and the capacity is about 35 tons per day, at an expenditure of about 2½ H.P. The ore is not roasted dead, however, in this case, about 6 per cent. of sulphur remaining in it. No less than twenty-three of these furnaces, with Brown and Allen's improvements, were in operation in the United States in 1893.[*]

Spence Furnace.—This consists of a series of four or five superposed hearths, communicating with one another by means of vertical passages at alternate ends of the furnace. The rabbling is done by rakes having a reciprocating motion longitudinally in the furnace and armed with teeth of triangular section having the apices pointing in the opposite direction to that in which the ore travels. When the rakes move in the direction in which these apices point, the ore is only stirred, but when the return movement is made, the flat sides of the teeth push some of the ore along the floor of the furnace, and a part falls through the vertical passages on to the lower hearths. The ore is charged in through a hopper and discharged roasted into a pit. The rakes are withdrawn at intervals to allow time for cooling. This furnace was tried at the Alaska Treadwell Mine, and subsequently abandoned in favour of ordinary hand-rabbled reverberatory furnaces.

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[*] *Eng. and Mng. Journ.*, May 20, 1893, p. 463.

One of the chief causes of difficulty and expense in working furnaces with mechanical stirring apparatus is that the iron hoes gradually become heated, and they are then rapidly corroded by the sulphur in the ore. In hand stirring the rabbling tool is withdrawn as soon as it is hot, and allowed to cool, while another is substituted for it meanwhile, thus prolonging its life. In furnaces with mechanical rabbles the trouble and expense caused by the wearing of the hoes were very great until the device was adopted of placing the moving parts of the mechanism outside the ore-hearth with the stirring hoes passing through a slot in the furnace wall. This device is used in some of the furnaces described below.

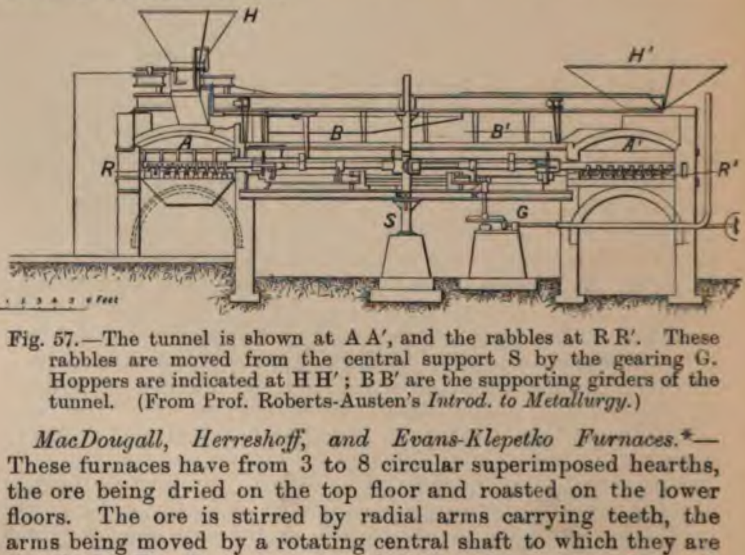

Fig. 57.—The tunnel is shown at A A′, and the rabbles at R R′. These rabbles are moved from the central support S by the gearing G. Hoppers are indicated at H H′; B B′ are the supporting girders of the tunnel. (From Prof. Roberts-Austen's *Introd. to Metallurgy.*)

*MacDougall, Herreshoff, and Evans-Klepetko Furnaces.**—These furnaces have from 3 to 8 circular superimposed hearths, the ore being dried on the top floor and roasted on the lower floors. The ore is stirred by radial arms carrying teeth, the arms being moved by a rotating central shaft to which they are attached. The furnaces are made hot and subsequently work without fuel, reducing the sulphur in concentrates at Butte to about 4 or 6 per cent.

Pearce Turret Furnace.—This consists of an ordinary reverberatory hearth built in an annular form (see Fig. 57). In the centre of the circular space surrounded by the hearth is a vertical iron column carrying four hollow horizontal arms projecting through a slot into the reverberatory hearth which they cross transversely. The column revolves and the arms carry rabble blades which traverse the hearth, stirring the ore and moving it round the circle by degrees. "Air is forced through the hollow arms and is discharged against the rabble blades, performing the double duty of cooling the iron work and of furnishing heated air for the oxidation of the ore." The ore is discharged automatically after passing once round the furnace. Two or more fireplaces are used. The hearth is 6, 7, or 8 feet wide, and is now sometimes built in two and even six floors, one above the other. Moreover, the top of the arch is sometimes used as a drying hearth. An improvement on the air blast for cooling the rabble arms consists in supplying them with water-jackets, by which their life is lengthened, the air being let in through the fire-boxes and the apertures in the outer wall. Moreover, in imitation of the intermittent movement in the Brown furnace, a single arm is now used with some pyritic ores, running round the hearth at intervals and stopping automatically for a specified period in the open space above the discharge hopper.

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* "Notes on the Metallurgy of Copper of Montana," by H. O. Hofmann *Trans. Am. Inst. Mng. Eng.*, Feb. 1903.

The Brown Horse-shoe Furnace.—In this furnace, varieties of which are called the "Elliptical" and the "Parallel," the ore-hearth (L, Fig. 58*b*) has a narrow chamber, M, on each side of it and separated from it by tiles projecting downwards from the arch and upwards from the hearth, so that only a narrow slot is left between the two series. In the lateral chambers, M, are laid rails on which the carriages supporting the stirrers are moved by a cable running continuously on grooved wheels. These are placed in doorways in the wall of the furnace (see Fig. 58*c*) to keep them cool. The carriage spans the ore-hearth, passing through the slots connecting L and M, and is arranged to grip the cable or release it. The carriages support steel stirrers which just reach the ore-hearth and stir and move on the ore. In operation half the carriages are traversing the furnace while half are resting in a cooling space (O, Fig. 58) outside the furnace. When two carriages are used, the one moving out of the furnace strikes the other, which is at rest in the cooling space, and pushes it forward until it grips the cable automatically and starts off to take up the stirring, and the hot moving carriage is at the same time automatically released and comes to rest. The stirrers are thus prevented from becoming overheated.

There are one, two, or more fire-boxes (P, Fig. 58) according to requirements, and there is generally a prolongation of the ore-hearth outside the furnace walls, used as a cooling hearth, over which the ore is pushed by the stirrers, until it falls into a hopper ready for chlorination. Fig. 58 is a plan of the furnace erected in 1896 at the Golden Reward Chlorination

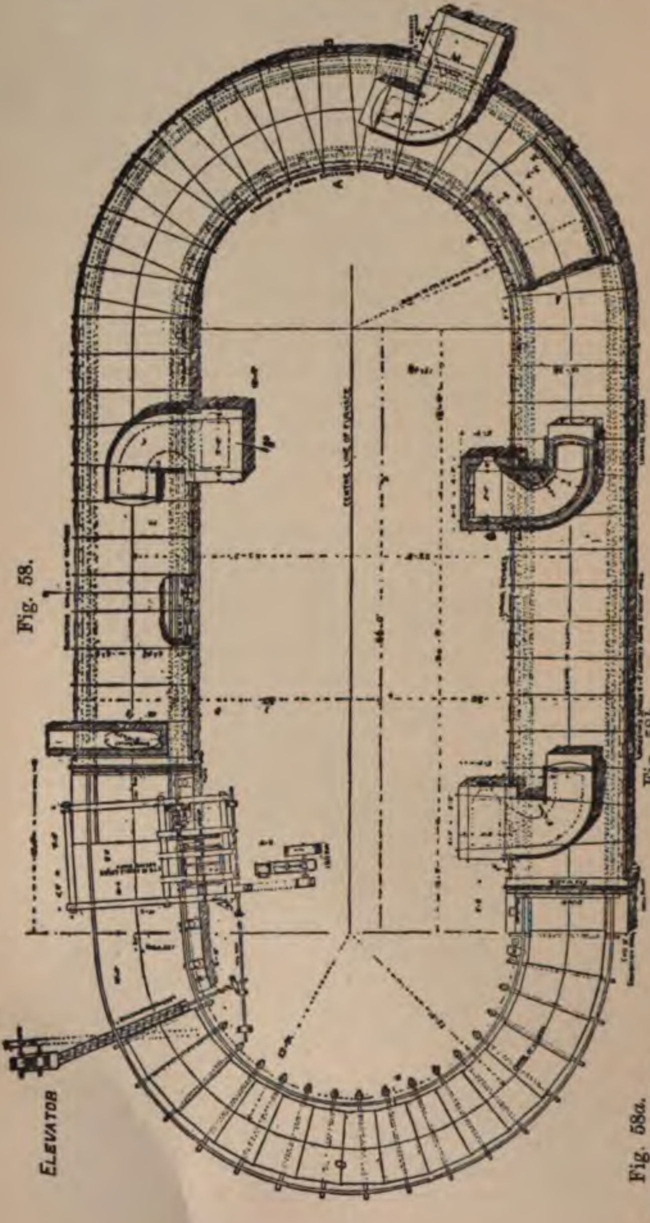

Fig. 58.

ELEVATOR
CENTRE LINE OF FURNACE
A

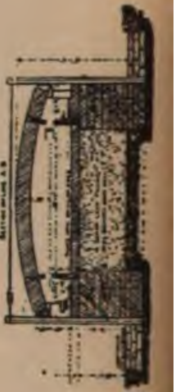

Fig. 58c.

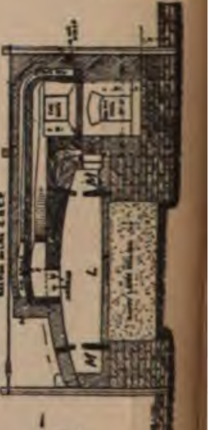

Fig. 58b.

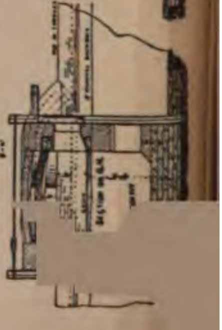

Fig. 58a.

Mill, at Deadwood, Dakota; Fig. 58*a* is a longitudinal section through one of the fire-boxes; Fig. 58*b*, a transverse section through the ore-hearth and fire-box, and Fig. 58*c*, a transverse section through a part of the furnace where there is no fire-box. The scale is $\frac{1}{20}$ in. per foot in Fig. 58 and $\frac{1}{10}$ in. per foot in the others. In the Golden Reward furnace the roasting hearth is 180 feet long and 8 feet wide, and the cooling hearth 78 feet long. Three rabbling carriages are used, following one another at intervals of 90 seconds, and moving the ore forward at the rate of 22 feet per hour. The ore contains from 2½ to 8 per cent. of sulphur, and it is stated that 65 tons of ore crushed through 30-mesh sieves are roasted per day, the product containing 0·3 per cent. of sulphur. The capacity is here 1 ton to every 22 square feet of hearth per day, and the total cost is less than 50 cents per ton for roasting, cooling, and conveying the ore to the barrels,* the cost per day of twenty-four hours being made up as follows:—

Fuel, 6 cords of wood, ...	$19.50
Labour, two men, one each shift, ...	5.00
Power, 5 H.P., ...	1.25
Oil, lights, repairs, &c., ...	1.50
	$27.25

The Brown furnace is also able to treat concentrates, and several are at work in various mills in Western America.

The Ropp Straight-line Furnace differs from the Brown furnace in having the cable placed underneath the hearth, the slot being in the hearth itself. Through the slot an arm, connected below with a four-wheeled truck, extends upwards, and across the upper end of this arm is the supporting bar from which the stirring teeth project. The chamber beneath the hearth is easily accessible for repairs, and the trucks and rabbles are cooled outside the furnace while traversing the track leading back to the starting point. The hearth of the standard furnace is 105 feet long and 11 feet wide, and its capacity seems to be about the same as the Brown furnace of equal area. Several of these furnaces are in use in Western America.

The Holthoff-Wethey Furnace † is of similar design. Six of these were used at the Great Boulder Perseverance Sulphide Mill in 1901. The fixed horizontal hearths were 120 feet long and 12 feet wide, the capacity of each furnace being 60 to 70 tons per day. The ore, which apparently contains about 3 or 4 per cent. of sulphur, is stirred and carried forward by means of eight travelling rabbles in each furnace, and less than 0·02 per cent. of sulphur in undecomposed sulphides is left in the ore after roasting.

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* *Eng. and Mng. Journ.*, July 4, 1896.

† Charleton, *Gold Mining and Milling in Western Australia*, p. 348; *Mineral Industry for 1897*, p. 448; and "Notes on Metallurgy of Copper," by Hofmann, *Trans. A.I.M.E.*, Feb. 1903.

The Richards Furnace[*] was introduced at Mount Morgan and subsequently installed at the Great Boulder Main Reef Sulphide Works. It consists of eleven superimposed hearths, the whole furnace being 30 feet long, 12 feet wide, and 65 feet high. It is heated by a single fire, and the issuing gases are stated to be at a temperature of 300°. The ore is not stirred, but banks up on each hearth until it forms a natural slope when it slides down, falling from floor to floor through vertical flues. At Mount Morgan the cost of roasting in this furnace was put at 5s. per ton.

The Edwards Furnace[†] consists of an iron frame lined with brick, and is usually about 60 feet long and 6 feet wide inside. There are 15 rotating rabbles moved by shafts passing through the arch and actuated by bevel gearing. A considerable number of these furnaces are in use at Kalgoorlie, the amount of ore roasted in each being from 12 to 18 tons per day.

The Merton Furnace[‡] consists of three superimposed hearths and a fourth hearth in front of them and at a lower level. The main hearths are 23 feet 9 inches long and 8 feet wide over all. The height of each arch is 16½ inches in the middle and 9 inches at the side. The ore is charged in on the upper hearth, stirred and moved forward by rotating rabbles attached to vertical shafts, and delivered in succession to the middle and lower hearths and finally to the finishing hearth. There are four shafts, with a rabble attached to each shaft on each of the three floors. The hearths are horizontal, having no fall. The capacity on Kalgoorlie sulpho-tellurides is said to be from 18 to 25 tons of ore per day.

Revolving Cylindrical Furnaces—*The Brückner Cylinder.*—This furnace was first introduced in Colorado in 1867, and is used chiefly in the chloridising roasting of silver ores, although it is also suitable for dead roasting. It consists (Fig. 59) of a cylinder of boiler-plate iron, lined with firebricks. The ends are partly closed, leaving central apertures 2 feet in diameter. There are also four receiving and discharging openings (*a*, *a*), closed by hinged doors. The discharge is effected into a pit, or into hot-ore cars placed underneath. The cylinder revolves on two chilled-iron friction rings (*b*, *b*), resting upon four chilled-iron carrying rollers (*c*, *c*). The rotation is caused by friction between the rings and the rollers, which are driven by a belt and pulley. The older method of turning the cylinder by means

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[*] Charleton, *op. cit.*, p. 326.

[†] Evan Simpson, *Trans. Inst. M. and Met.*, Oct. 15, 1903; *Eng. and Mng. Journ.*, Feb. 11, 1904, p. 242.

[‡] *Australian Mines and Mining*, by Donald Clark, p. 21. *Eng. and Mng. Journ.*, November 21, 1904, p. 775. Evan Simpson, *loc. cit.*

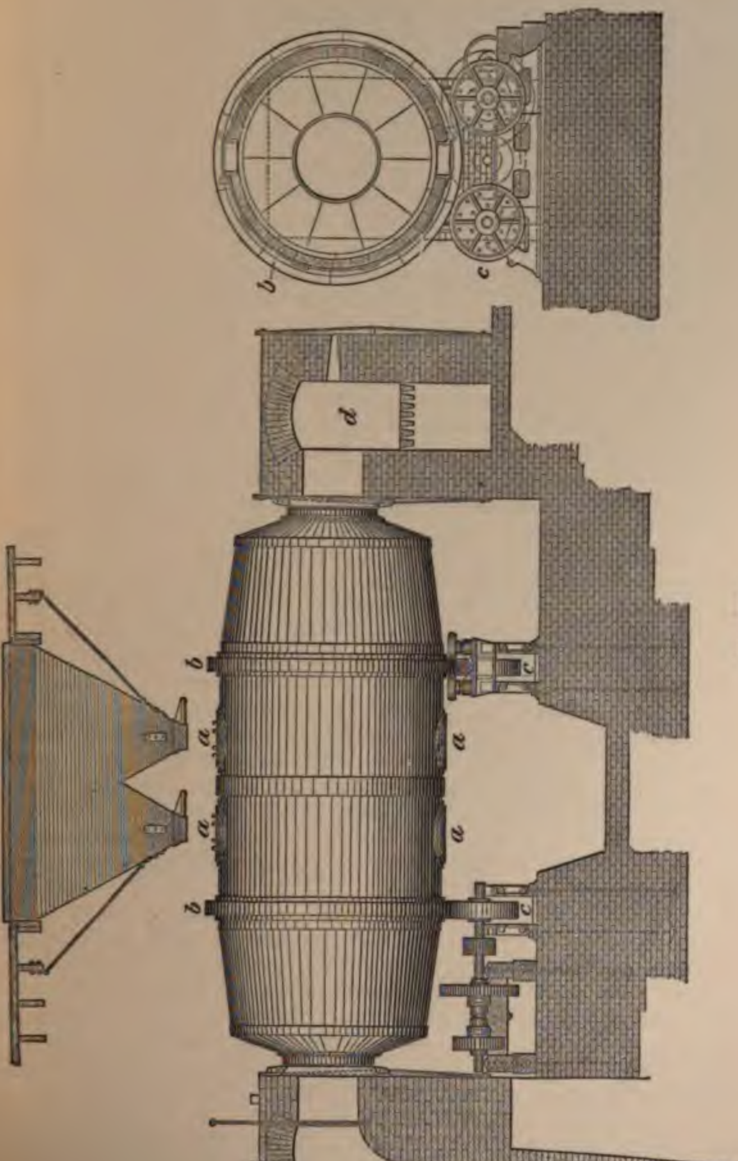

b
c
p
b
a
a
b
c
a
a
c

Fig. 59.

of a gear-wheel, connecting with teeth set on the cylinder, has been abandoned.

The lining is one brick (2½-inch) thick in the middle of the cylinder, but additional layers are added near the ends until the circle is contracted down to the size of the openings in the ends, which are also lined; each layer falls short of the preceding one by about 2 inches, so that the lined ends have a conical form. The mortar used consists of one-third fireclay and two-thirds crushed firebrick. In the old form six iron pipes passed through the cylinder in a plane at an angle of 15° to the axis, and perforated plates uniting them formed a diaphragm which assisted in stirring the ore. The circular flange surrounding the opening at one end of the cylinder connects loosely with the fire-box, *d*; the other end connects with an opening leading to dust chambers and the stack. The dust chambers (not shown in the figure) are large, and must be carefully attended to, as the amount carried over is usually 10 per cent. of the ore, and may rise to 25 per cent. when the mineral is light. The heavier particles collected in the first chamber are charged into the furnace again; the lighter dust is unsuitable for this purpose, and must be subjected to special treatment. In the improved Brückner the diaphragm is discarded, as it is rapidly corroded by the sulphur, and increases the quantity of material carried into the dust chambers. The furnaces are now usually made 7 feet in diameter and 18 feet long, a full charge being from 6 to 8 tons. This large size has given greater satisfaction than that formerly employed—viz., 6 by 12 feet—the full charge of which was from 3 to 4 tons of ore. As a charge can be kept in the furnace as long as is necessary, the Brückner is particularly useful when very base ores are to be roasted, or when the ores vary greatly in composition, so that the duration of roasting is not constant. The mixing of the ores is rendered more perfect by the conical shape of the ends, which causes the ore to be thrown backwards and forwards, changing its position frequently and exposing new surfaces to the action of the fire.

Brückner's cylinder has been found suitable for the dead oxidising roasting of pyritic ores, a little charcoal being sometimes added towards the end of the operation to reduce the sulphate of copper formed. It has the disadvantage of being too hot at first, and not hot enough at the finish if the fire is kept uniform, and consequently pyritic ore tends to form into balls. These may contain sulphates, which are not then easy to decompose except with great waste of time and fuel. The consumption of fuel is also rendered greater by the shortness of the furnace, as a large amount of waste heat passes out into the dust chamber, and is not utilised, a state of things exactly similar to that which occurs in the reverberatory furnace with only one floor.

Hickock's improvement in Brückner furnaces was as follows:— Between the fire-box, which is mounted on wheels, and the revolving cylinder is placed a short iron cylinder, 18 inches long, rigidly connected with the fire-bridge and closely fitting the throat of the revolving cylinder. In the lower side of the small cylinder is an aperture, from 8 inches to 10 inches square, closed by a sliding door. This is useful in regulating the draught, without cooling down the fire. By its use a sheet of cold air can be made to pass immediately above the ore and below the products of combustion of the fire. It is stated that a considerable saving of fuel has been effected since the introduction of this simple device, and that a more perfect roast can be obtained in one-third less time than was formerly possible.

The Hofmann Furnace.—This furnace was designed by H. O. Hofmann as an improvement on the Brückner cylinder, which has the disadvantage of being much hotter at one end than at the other. The result of this is that the ore near the fire is exposed to a higher temperature than that near the flue, and, consequently, it is finished much sooner, so that fuel is wasted while finishing the ore in the colder part of the furnace, and, in the case of antimony ores, if the temperature is high enough to roast the ore next the flue, the portion close to the fire becomes caked and suffers a considerable loss of silver by volatilisation. The Hofmann furnace (Fig. 60) has a fireplace and flue at each end. The flues are between the fireplaces and the cylinder, and open downwards into dust chambers, C, built beneath, which are connected with the main stack. The arrangements at each end are the same, and are such that, by means of dampers, the current of the air and gases can be made to pass through the furnace in either direction. The fire is lighted first at one end, A, and the dampers arranged so that the draught passes through the revolving cylinder and down the flue, B, at the other end. After a few hours the fire is lighted at the other end, in the fireplace, D, and the position of the dampers is reversed. The alternate heating from the two ends is regularly performed until the charge is completely roasted. In this way more uniform heating is obtained, both halves of the charge being raised to the required temperature without any portion being overheated. It is stated that the formation of balls is diminished by this system, and, in particular, the furnace is found to answer well when treating ores which require either a very low roasting temperature, or a very high one. Thus, for example, antimonious ores, which cake readily, are said to be successfully treated by the use of moderate fires; and ores containing very little sulphur, and so requiring a higher temperature, can readily be made as hot as is necessary by the alternate use of large fires, without using much fuel. When one fire is in operation the other is allowed to go down. By closing one of the large dampers near the main flue, and opening the damper of the corresponding descending flue, and the "plug door" in connection with it, a current of fresh air can

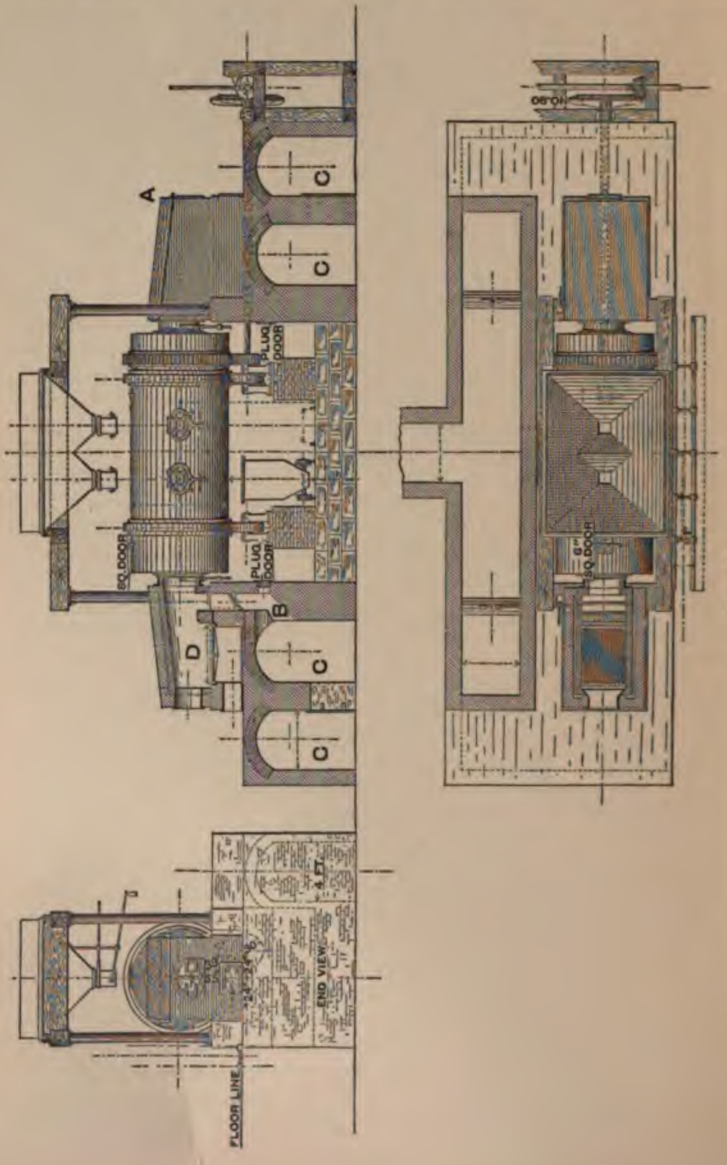

A
B
C
C
C
C
D
PLUG DOOR
PLUG DOOR
SQ. DOOR
6" SQ. DOOR
END VIEW
4 FT.
FLOOR LINE

Fig. 60.

be introduced into the furnace beneath the flame from the fire, thus greatly increasing the rate of oxidation of the ore. This arrangement is especially useful when ores carrying a high percentage of sulphur are being treated, their tendency to form balls being in this way greatly diminished.

It is clear that the use of much larger cylinders than those of the Brückner furnace is rendered possible by the presence of a fire at each end. Nevertheless, the evil of heating the ore unequally is only palliated; the ore in the middle of the furnace is never heated so strongly as that at the two ends, whilst in the case of continuous-discharge inclined cylinders, every particle of ore is heated similarly. Moreover, the use of two fires, which are alternately checked and urged, must cause a considerable waste of fuel. The principle is obviously inferior to that of the White-Howell furnace.

One incidental advantage of this furnace over all other revolving cylinders lies in the absence of loss from the carrying away of dust at the moment of charging-in. In other cylinders the ore falls in a shower into the furnace through a strong draught, by which much of the dust is carried off into the flue. When charging the Hofmann furnace, however, the dampers in both the descending flues are left open, so that the draught passes through both fireplaces direct to the dust chambers, without entering the furnace at all.

The White and White-Howell Roasting Furnaces.—The *White furnace* consists of a long cast-iron revolving cylinder, lined with firebrick, and inclined towards the fire end. The cylinder is bound by four cast-iron rings which rest on friction wheels, serving as supports, and it is revolved by other friction wheels driven by a shaft and pulleys. Crushed ore is fed into the furnace continuously at the upper end, passes through it by gravity, and is continuously and automatically discharged by falling into a pit through an opening in the floor close to the fire. The time occupied by the ore in passing through the furnace depends on the angle of inclination of the cylinder which can be changed, so that the time of roasting can be shortened or lengthened according to the nature of the ore. Ores containing a high percentage of sulphur require to be subjected to heat for a longer time than those with little sulphur, and the angle of inclination is reduced in such cases. The average inclination is about 1 in 20. The cylinder is lined with firebrick throughout, and projecting bricks raise portions of the ore and drop it through the flames, thus assisting the oxidation.

The advantages claimed for this furnace are that it is continuous in its operation, discharging its product regularly into the pit at the lower end, and this roasted ore can be allowed to accumulate and be withdrawn as required. The ore is sub-

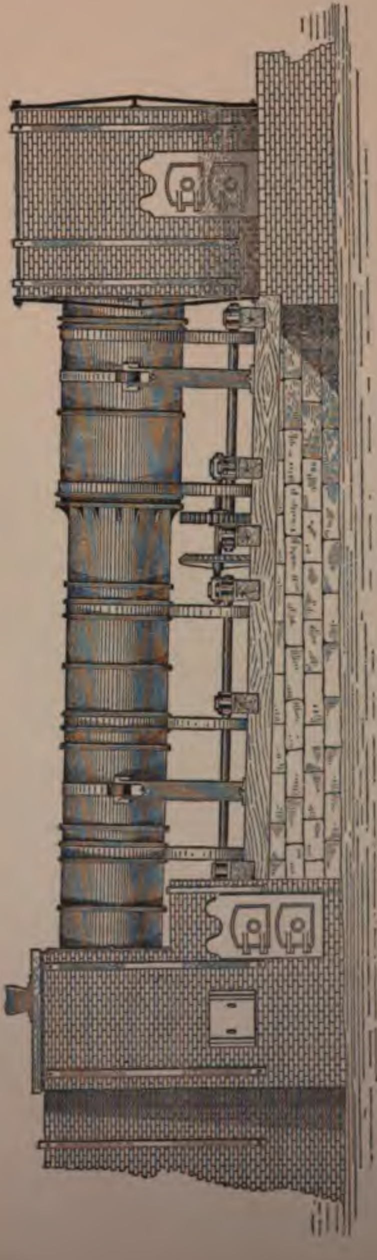
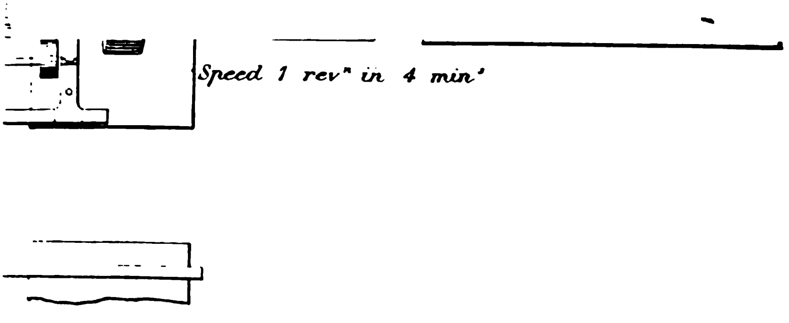


Fig. 61.

mitted to a gradually-increasing temperature, the most favourable conditions for dead roasting being thus obtained. The usual size of this furnace is about $4\frac{1}{2}$ feet in diameter and 27 feet long, the capacity being usually stated at from 20 to 30 tons per day. When employed for the dead roasting of pyritic ores its capacity is less, and in some cases the passage through a 27-foot cylinder is insufficient to eliminate the whole of the sulphur. In such cases the ore is made to traverse successively two similar cylinders, the combined length of which is sometimes over 60 feet. Great care should be exercised in keeping the ore supplied to the furnaces as uniform as possible, so that when once the rate of feed and the proper angle of inclination for the ore have been determined, no further alterations are needed in order to continue to give a perfectly-roasted product.

In the *White-Howell furnace* (Fig. 61) only the enlarged part next the fire-box is lined with fire-brick, the remainder being left unlined. Cast-iron spirally-arranged shelves assist in raising and showering the pulp through the flames. To both White and White-Howell furnaces an aux-



Speed 1 revⁿ in 4 min'

iliary fire is often added for roasting the dust which escapes from the main furnace. The dust, when it has been completely roasted, is shovelled out and mixed with the main bulk of the ore by hand. This arrangement is decidedly inferior to that suggested by John E. Rothwell,* who uses a hopper-shaped dust chamber with its bottom consisting of an inclined cast-iron plate projecting about 8 inches into the upper end of the cylinder. The dust carried out of the cylinder settles in this chamber, and, as it accumulates, slides down the sides and mixes with the fresh ore. The ore is thus kept more uniform than if re-mixed by hand, and some saving in labour is also effected. Rothwell uses a cylinder 36 feet long and 5 feet in diameter with an inclination of 14 inches only, rotating once per minute. The lining is of firebrick, 6 inches thick.

The Argall Roasting Furnace.—This is a modification of the older forms of rotating cylindrical furnaces, and consists of four small parallel tubes instead of one large one. The tubes are about 28 feet long, 2 feet in diameter, and lined with firebrick, which is thicker at the lower end nearest the fire. The ore is equally distributed between the tubes, so that the furnace is balanced and requires but little power to rotate it. The furnace is rotated by friction drivers, and makes about one revolution in four minutes, the slowness of movement of the ore diminishing the loss by dusting. The working drawings reproduced in Plate V., were supplied for the purpose by Mr. Philip Argall. The furnace as shown is arranged for burning coal, but some have been erected and were used for some time with oil as fuel. The capacity of the furnace when roasting ores containing 2 per cent. of sulphur is from 45 to 50 tons per day, the sulphur being reduced to 0·1 per cent.; the ore is from two to three hours in the furnace, and the consumption of slack coal for this ore is 150 to 160 lbs. per ton. On pyritic ores containing 12 per cent. of sulphur, the capacity is said to be from 25 to 30 tons per day. The cost of roasting is very small, hardly any repairs and supervision being necessary. The furnace has been used continuously for some years at Cyanide, Colorado, in roasting Cripple Creek ores for cyanide treatment. Similar furnaces with modifications, are used for drying ore and also for cooling roasted ore. In the cooler, water tubes and jackets reduce the temperature of 300 tons of ore per day from a red heat to the ordinary temperature.

Use of Producer Gas in Roasting.—The use of producer gas in roasting may be mentioned. It was introduced at the Holden Mill, Aspen, Colorado, in 1891, and soon completely displaced other fuel there for both drying and roasting.† The gas plant consisted of two "Taylor" revolving-bottom gas producers, one 6 feet and the other 7 feet in diameter. The coal used was a mixture of one-third "Sunshine" nuts and two-thirds "Newcastle" nuts and pea-coal, the analyses of which were as follows:—

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* *Mineral Industry for 1892*, p. 234.

† W. S. Morse in *Trans. Am. Inst. Mng. Eng.*, Montreal meeting, 1893.

	Sunshine.	Newcastle.
Water, ...	2·8	1·7
Volatile matter, . .	36·3	37·95
Fixed carbon, . .	37·1	48·6
Ash, ...	23·8	11·6
	100·0	99·85

The drying plant consisted of four Stetefeldt double shelf driers, and the roasting furnace was a Stetefeldt shaft furnace. The ore contained 8·10 per cent. of sulphur before roasting and 0·2 per cent. as sulphide afterwards. The coal consumed was 72·22 lbs. per ton of ore dried and 117·44 lbs. per ton roasted, the cost for the two being 29·37 cents per ton of ore. All the above figures are averages for a run of over a year—viz., from November, 1891, to January, 1893. The heat obtained in the Stetefeldt furnace could be made sufficient to fuse the ore; and there is, therefore, no doubt that similar producers could give heat enough to desulphurise ores in reverberatory furnaces.

Producer gas was introduced at the Great Boulder Sulphide Mill, Kalgoorlie, in 1901 * for use in the Edwards furnaces. It is produced from a mixture of coal and sawdust. There are three Dowson producers, 15 feet high and 6 feet in diameter. The gas is delivered to each furnace by a 6-inch pipe, from a 20-inch main, and distributed to various parts of the furnace by smaller pipes. The use of wood for firing was apparently not entirely dispensed with at first.

Cooling Roasted Ore.—This was formerly effected by spreading the roasted ore on sheet-iron floors, and either leaving it for twenty-four hours, or damping it down by a hose-pipe. More recently † it has been passed through plain or multitubular cylinders cooled with water, or passed back by automatic rabbles above or below the stationary hearth furnaces, or allowed to fall from shelf to shelf through air in towers built on the Hasenclever principle.

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* A. G. Charleton, *Gold Mining and Milling in Western Australia*, p. 336.

† *Mineral Industry*, vol. ix. (1901), p. 364. Article by J. E. Rothwell.

CHAPTER XIII.

CHLORINATION: THE PLATTNER PROCESS.

Introduction.—The use of chlorine as an agent for the extraction of gold from ores was first suggested by Dr. John Percy, F.R.S., at the Swansea meeting of the British Association, held in August, 1848, in a paper* embodying the results of experiments carried out in the year 1846. Simultaneously, in 1848, Prof. C. F. Plattner, Assay Master at the Royal Freiberg Smelting Works, applied chlorine gas to the assay of the Reichenstein residues, and proposed that a similar method of treatment should be adopted on a large scale. These residues were the result of treating the Reichenstein ore with the object of extracting the arsenic. They consisted chiefly of oxides of iron and oxidised arsenical compounds, and had been roasted in the course of the process for the extraction of the arsenic. The residues had been accumulating for more than a century, and contained from 15 dwts. to 1 oz. of gold per ton; they were considered too poor to smelt, while they could not be made to yield the gold contained in them by amalgamation.

Prof. Plattner's suggestion was followed up by investigations made by Dr. Duflos in 1848,† and by Lange in 1849.‡ Dr. Duflos compared the results obtained by treating the residues with chlorine water by percolation in a stationary vat, and by agitation in a revolving barrel; and as these results were the same, he recommended the stationary vat as being more economical. He also obtained identical results with chlorine-water and with dilute solutions of chloride of lime and hydrochloric acid mixed together. On the other hand, Lange believed that gaseous chlorine, applied to the ore in the same manner as had been used by Plattner in assaying, was a more efficient agent than a solution of chlorine in water, and it seems to have been in accordance with his advice that the first chlorination works, that at Reichenstein, was established in 1849. The chlorinating vessels were small earthenware pots, and the precipitant employed was sulphuretted hydrogen. Plattner subsequently recommended wooden vats coated with pitch, and ferrous sulphate as a precipitant, and although these were not at first used at Reichenstein, they were adopted by G. F. Deetken in 1857, when he introduced the system into California.

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* *Phil. Mag.*, 1853, vol. xxxvi., pp. 1-8.

† *Erdm. Journ. Prak. Chem.*, vol. xlviii., 1849, pp. 65-70.

‡ Karsten's *Archiv*, vol. xxiv., 1851, pp. 396-429.

The following three systems of chlorination have been developed, and are all in use in different parts of the world :—

1. The *Plattner process*, in which the gold is dissolved by means of gaseous chlorine acting on moist ore. The actual solvent is really a saturated solution of chlorine in water.

2. The *Barrel process*, in which a supersaturated solution of chlorine in water is used, the chlorine being kept in solution by an atmosphere of chlorine of a few pounds pressure inside the barrel.

3. The *Vat-solution process*, in which weaker, unsaturated solutions of chlorine are used.

These processes are described separately in successive chapters, in the order of their introduction on a working scale.

Plattner Process at Reichenstein.—The following description of the original process employed at Reichenstein is an abstract of the account given in Kerl's *Hüttenkunde*, vol. iv., p. 372, 1865. The material treated consisted of the residues obtained by roasting arsenical iron pyrites for the production of white arsenic, which was volatilised and condensed in brick chambers. There were forty-eight earthen chlorination pots, each holding 150 lbs. of ore. These pots were strengthened with iron hoops, and suspended on two journals, so that they could be discharged by inverting them.

The lower part of the pots was of a conical shape, and this part was filled with pebbles and sand covered by a perforated earthen plate, the function of which was to prevent the ore from mixing with the filter bed. The ore filled the cylindrical part of the pot above the earthen plate. The chlorine was generated by the action of hydrochloric and sulphuric acids on manganese dioxide in earthenware vessels, and was conveyed thence to the ore-pots through leaden pipes. The gas was introduced below the filter bed, and passed upwards through the ore for an hour; a wooden cover was then fitted on, but not luted down until chlorine had been passed for from six to seven hours longer, after which all joints were luted down with dough, and the vat left until the next day. The cover was then removed and water, at a temperature of from 64° to 77° F., poured on, and allowed to percolate through the ore and filter bed by gravity. The liquid coming from twenty-four pots was conveyed to four vats, the first one being filled with solution before the second was used, and so on; the contents of the fourth vat, being too poor for precipitation, was used over again for leaching. The leaching was stopped when 90 cubic feet of water had passed through the total charge of 3,600 lbs., this being at the rate of 312 gallons per ton of 2,000 lbs. The liquid from the first three vats was drawn off into twenty glass globes, which were heated on a sand bath so as to raise the temperature to 77° F. Sulphuretted hydrogen, obtained from fused sulphide of lead and sulphuric acid, was passed through until the saturation point was reached, when the liquid was left to settle until the next day; after this the clear supernatant liquid was passed through sawdust filters to catch all the sulphide of gold, which might still have been in suspension. The sulphides were refined by dissolving them in acids, precipitating the metallic gold with ferrous sulphate, and melting it in clay crucibles with nitre and borax. The amount of arsenides treated daily was 3,600 lbs., containing about $\frac{5}{9}$ oz. of gold per ton, so that only about 250 ozs. of gold were extracted yearly, and it is difficult to believe that the enterprise could have been a commercial success.

Modern Practice in the Plattner Process.—The system ascribed to Plattner consists of the following operations:—

1. The concentrates or residues are subjected to a "dead" roast in a reverberatory furnace.

2. The roasted ore is slightly damped with water and charged into wooden vats, holding from 1 ton upwards. The vats have false bottoms consisting of filter beds of gravel or of cloth. Chlorine gas, generated in another vessel, is introduced at the bottom of the vat, and rises through the ore, permeating every part of it. The vat is then closed up and left undisturbed for twenty-four hours or more, by which time all the gold is converted into soluble chloride of gold.

3. The soluble salts are then washed out by water, which is allowed to flow on to the surface of the ore, and, passing through it, drains through the filter bed. When all the gold has thus been removed in solution, the tailings are thrown away.

4. The solution of gold chloride is acted on by ferrous sulphate, or some other suitable reagent, by which the gold is precipitated; the particles of the precious metal are allowed to settle, and then are collected and melted down.

This process is still used for the treatment of small quantities of roasted concentrates in a number of mills. The principal modifications which have been made in it have been introduced mainly in order to deal with larger quantities of material.

The Vats.—The vats used for impregnating the ore with chlorine are usually, in California, about 7 feet in diameter, and are made of staves 3 feet long and 2 inches thick, which consist of the best split sugar-pine. They are coated with a mixture of pitch and tar to protect them from the corrosive action of the chlorine. Before being used for the first time, the vats are thoroughly soaked with water to diminish their absorptive action on the chloride of gold solution, but all wood brought in contact with this solution is nevertheless invariably impregnated with a certain amount of gold. This may be recovered after the vats, &c., are worn out, by burning them and fusing the ashes with suitable fluxes. The false bottom generally consists of quartz pebbles, the lowest layer being of the size of hazel nuts, and each successive layer consisting of finer material until, at the top, a thin layer of fine sand (passing a 20-mesh sieve, but retained on a 60-mesh sieve) is spread evenly over the surface. The thickness of the filter bed (which is not shown in the figure) is usually from 6 to 12 inches. It is supported on boards (A, Fig. 62), 1 inch thick, in which numerous ½-inch auger holes are drilled; these boards rest on wooden strips (not shown in the figure), 3 inches wide and 1 inch thick, which do not reach the edge of the vat, and so keep a clear space 1 inch deep, just above the true bottom of the vat, in which the solution can accumulate. The solution is drawn off by a leaden pipe fitted with a stopcock, preferably of stoneware; the pipe should be level with the bottom of the vat, which may with advantage be made with a slight fall towards the outlet to prevent any liquid being left in it. Deetken states[*] that fine sea-shells (consisting of carbonate of lime) have been used instead of quartz pebbles for the filter beds without any prejudicial result. Talcose rocks, and particularly silicates of alumina, must not be used on account of their power of absorbing the chlorine. For the same reason sulphides, magnetic iron oxide, metallic iron, fragments of wood or other organic matter, or, briefly, any substances capable of being acted on by chlorine or of reducing the chloride of gold must be carefully excluded from the filter bed.

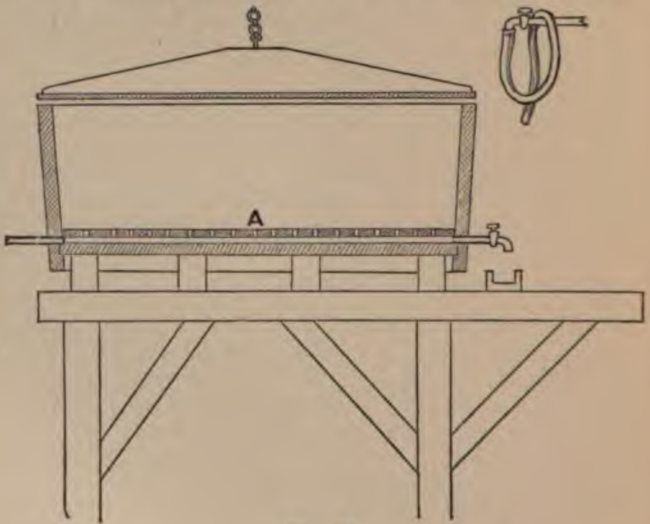

Fig. 62.

The surface of the filter bed may be covered with boards, not fitted closely together, but made into a framework by cross-pieces and pierced with many auger holes. This cover is useful when the tailings are being cleared out, otherwise, in shovelling away the ore, the surface of the filter bed is partly removed also. Similar devices used in the cyanide process to avoid this are given in Chap. xvi. Filter cloths of canvas, burlap, or cocoa-nut fibre matting are also frequently used above the filter-bed, stretched tightly over a framework of wood which accurately fits the inside of the vat. The space between the canvas and the wall of the vat is packed with hemp or other material closely tamped down. Filter-cloths of every material, except asbestos, are soon rotted by the action of the chlorine, and their use is frequently dispensed with. Wool lasts longer than cotton.

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[*] *Mineral Resources of the Rocky Mountains*, 1873, p. 342.

Charging-in the Ore.—When the vats are ready to be charged, a layer of dry ore is spread over the false bottom, and time given for the water from the filter bed to be drawn up into this layer by capillary attraction. If attention is not paid to this point, the lowest layer of ore becomes too wet from the combined effect of the water added to it before charging-in and that absorbed from the false bottom. The result is that the passage of the chlorine through the mass is resisted, and there is a great increase in the consumption of the gas. Deetken states that the whole of the usual charge of gas may be thus consumed, not rising more than a few inches above the bottom. The greater part of the charge is damped by sprinkling with water and thorough mixing. The amount of the water added varies with the nature of the ore, but the usual amount is from 6 to 12 per cent. for roasted ores. If it is made too wet, dry ore to the required amount is mixed with it. A good rough method of ascertaining when it is of the proper degree of dampness is to compress some in the hand; balls of ore should be readily formed in this way, but should be just dry enough to crumble up again. The reason for the addition of water is that perfectly dry chlorine has scarcely any action on metallic gold at ordinary temperatures, and up to a certain point an increase in the amount of water present raises the rate of solubility of the gold. The limit of the amount of water that can be added is, however, determined by physical conditions, as the mass must be of loose porous texture in order to permit the gas to permeate readily through every portion of it. In order to promote this porous texture and uniform dampness, the ore is shovelled upon, and made to pass through, a sieve of four holes to the linear inch. This sieve may be conveniently made to slide on rollers on iron rails placed above the vat, and the ore, shaken through it, falls into the vat in a light shower. Although left undisturbed as far as possible, the charge must be levelled off with a rake occasionally.

When the vat is filled to within 6 inches of the top, the surface of the ore is made concave or saucer-shaped, higher at the sides than in the centre. The cover, usually of wood, is then lowered on to the vat by means of a chain and pulley, and the rim luted with a mixture of wet clay and sand, or, more usually in former times, with dough. These joints are kept moist during the "gassing" process by wet rags. The gas is introduced through a lead pipe, which is shown on the left-hand side of Fig. 62, passing into the vat below the false bottom. A small hole is left in the cover through which the displaced air may escape, and the issuing gases are tested from time to time by means of a rag tied to a stick and moistened with dilute ammonia. As soon as chlorine is found to be coming off freely, the hole is plugged, but the current of gas is not stopped until after the lapse of one or two hours more, when the charge is supposed to be saturated with the gas, the total time required for the impregnation being usually from five to eight hours.

Generation of the Chlorine.*—The chlorine is generated in air-tight vessels of lead fitted with a stirring apparatus passing through the lid and worked from the outside. The gas necessary for a 3-ton charge of roasted concentrates may be generated in a leaden vessel of 20 inches in diameter and 12 inches deep. The joints of this generator and of all pipes traversed by the gas or by liquids carrying it in solution must be made by melting the lead, no solder being used except pure lead. The "burning together" of the joints is usually effected by an air-hydrogen or coal-gas blowpipe jet. The cover of the chlorine generator is made gas-tight by a water joint, 2 inches deep, as are also the apertures in the lid for the passage of the revolving stirrer (which is usually made of hard wood) and for the delivery tube. Heat is applied by placing the generator on a sand bath standing on a perforated arch over a fireplace. The sand bath is often replaced with advantage by a water bath, as the heat required is not more than 90° F., and sudden heating causes inconveniently tumultuous generation of gas. The charge for 3 tons of ore consists of 20 to 24 pounds of rock salt, 15 to 20 pounds of manganese dioxide, containing 70 per cent. of available material, and 35 pounds of oil of vitriol of 66° B., diluted with half its weight of water. The cover is usually removed to introduce the solids and the water, but the acid is added, half a gallon at a time, through a siphon. At Heywood's Works, California, the acid is contained in a lead vessel furnished with a glass stopcock, from which a small continuous stream is allowed to fall into the siphon.

The outlet tube is of lead, but connections in pipes are often made by short pieces of indiarubber-tubing, well greased on the

* See description and figure of the Edwards generator in the section on Miller's chlorine process, chap. xviii.

inside. These resist the action of chlorine fairly well. The gas is passed through the wash-bottle shown in Fig. 63. It is usually a large glass bottle or carboy with its bottom removed, supported in a lead-lined box filled with water. The gas is made to pass through about half an inch of water. The use of the wash-bottle is partly to free the chlorine from hydrochloric acid, or other impurities with which it is contaminated, but mainly to give an indication of the rate of flow of the gas; it is desirable that this should be as uniform as possible, as otherwise leakage is more difficult to prevent. As soon as the current of gas falls off, fresh acid is added and the vessel stirred. The wash-bottle, as usually constructed, would be quite inadequate to free the gas completely from hydrochloric acid, whilst, even if it were approximately eliminated, some more would be speedily formed by the decomposition of water by the chlorine. It is, therefore, fortunate that this elimination is not absolutely necessary, in spite of the customary declaration to the contrary which generally appears in the descriptions of the process. Thus it is frequently stated that hydrochloric acid will act on any sulphides left undecomposed in the ore, generating sulphuretted hydrogen which would precipitate the gold already dissolved in the impregnation tank. This statement would not need any criticism, if it were not for the fact that it has often been made, but has apparently never been contradicted. No doubt matters might be arranged for the above reactions to take place if sufficient care were taken, but in practice they are not to be feared for the following reasons:—

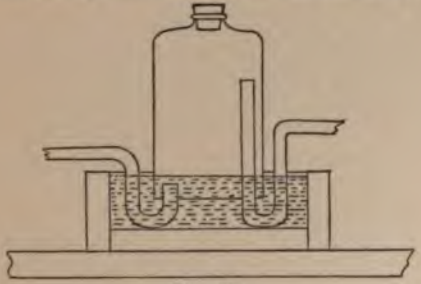

Fig. 63.
Scale, 1 in. = 1 ft.

(1) If undecomposed sulphides were present in the roasted ore, they would be attacked with much greater violence by chlorine than by hydrochloric acid, and before any gold could be dissolved, the whole of the sulphides present would be converted into sulphates, according to reactions, the final effect of which is shown in the following equation:—

$$R_2S + 4Cl_2 + 4H_2O = R_2SO_4 + 8HCl.$$

In some cases, no doubt, chlorides would be formed. (2) If the sulphides were not oxidised by the chlorine, they would be almost equally efficacious with sulphuretted hydrogen in precipitating the gold, so that even in this case hydrochloric acid would do no harm. (3) If an excess of chlorine is present, sulphuretted hydrogen could scarcely be said to be formed at all under any circumstances, as, if it were formed, it would be instantly decomposed by the chlorine. (4) If gold were precipitated by sulphuretted hydrogen in the impregnation vat, it would be in such a finely-divided state that it would be redissolved in chlorine in a very short time, provided the gas were in excess. The only disadvantage due to the presence of much hydrochloric acid in the gas lies in the fact that certain metallic oxides (oxides of iron, copper, &c.) are much more readily soluble in the acid than in chlorine, chlorides and water being formed, and the resulting solution will be contaminated with these chlorides, so that special precautions are necessitated to prevent the bullion from becoming base.

Impregnation of the Ore.—The ore is allowed to remain impregnated with the gas for from twenty-four to forty-eight hours, the continued presence of a strong excess of gas being ascertained at intervals by removing the plug from the cover and applying the ammonia test. When, as is usually the case, there is a large excess of gas when the impregnation is at an end, it may be disposed of in one of several ways. It may be dissolved by adding water before raising the cover (the usual method of procedure), or it may be withdrawn by aspiration and discharged outside the building, or stored in a gasometer for use in a subsequent charge. If the cover is raised before getting rid of part of the excess of the gas, the atmosphere of the mill is rendered unbearable for several minutes.

The time of impregnation varies according to the size of the particles of gold, the fineness of the metal, and the temperature employed. Chlorine has a very slow action on pure gold, the rate increasing gradually with the temperature up to 100°. The results of some experiments are given on p. 12. These results tend to show that both chlorine and bromine dissolve gold more rapidly at 50° to 60° C. than at ordinary temperatures, and that bromine is more rapid in its action than chlorine.

The fact that the rate of dissolution of gold depends on its state of division needs no demonstration. The influence of the fineness and chemical composition of the particles of gold alloy present in the ore was pointed out by Deetken and Küstel. Fine gold is acted on more slowly than that of low standard. The alloys containing base metals (copper, &c.) are dissolved very rapidly, and small quantities even of silver appear to increase the rate of solution, but if the percentage of silver amounts to 10 per cent. or over,* an insoluble coating of chloride of silver is formed over the granule, and further action is checked or completely stopped.

Reactions in the Impregnation Vat.—The amount of chlorine to be used depends mainly on the substances present, other than gold, by which chlorine is absorbed. A small amount is invariably converted into hydrochloric acid by the decomposition of the water present, the extent to which this reaction proceeds being increased by light and heat. If any sulphides are present they are oxidised by the chlorine in presence of water, sulphates and hydrochloric acid being formed, as follows:—

$$4Cl_2 + 4H_2O = 8HCl + 2O_2$$
$$CuS + 2O_2 = CuSO_4$$

These equations do not represent the whole of the reactions that take place. Some oxygen appears to be liberated, but the subject needs investigation. If considerable quantities of hydrochloric acid are thus formed, certain sulphates are converted into chlorides and sulphuric acid is set free, the reactions being influenced by the mass (*i.e.*, concentration) of the reagents present. Protosulphates or any other protosalts present are converted almost instantaneously to persalts by the chlorine, as follows:—

$$6FeSO_4 + 3Cl_2 = 2Fe_2(SO_4)_3 + Fe_2Cl_6$$

It is obvious from these reactions that great waste of chlorine in the impregnation vat is caused by imperfect roasting, 1 per cent. of unoxidised sulphur present in pyrites, if open to the attack of the chlorine, being enough to convert 8·9 per cent. of chlorine (or about 200 lbs. per ton of ore) into hydrochloric acid. It would, therefore, be a mistake to neglect to roast the ore dead and then try to retrieve the error by increasing the allowance of chlorine. Moreover, it demonstrates the uselessness of eliminating the hydrochloric acid from the chlorine before mixing it with the ore, and expecting in that way to prevent the ill effects produced by sulphides. The fact that many sulphides are almost instantly oxidised by even very dilute solutions of chlorine has been proved by a series of laboratory experiments by the author. The oxidation of protosalts is almost as rapid, although the percentage waste of chlorine is considerably less; thus 1 per cent. of sulphur present in the ore as ferrous sulphate will convert 1·1 per cent. of chlorine (or 24·6 lbs. per ton of ore) into hydrochloric acid.

Sulphate of copper ($CuSO_4$) does not appear to be acted on by chlorine, but, nevertheless, when much of it is present in a roasted ore, chlorination generally seems to be rendered impracticable. This is possibly due to the fact that some sulphate of iron accompanies it. Whenever sulphates of these metals are left in the roasted ore by accident or design it is necessary to remove them by a preliminary leaching with water before the chlorine is introduced. Of course if the ore is chlorinated in tubs by gas, it must be partially dried and sieved back into the tub before impregnation can be attempted.*

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* Hofmann and Magnuson, *Bull. Am. Inst. Mng. Eng.*, March, 1905.

Organic matter is also oxidised by chlorine, although much more slowly. At ordinary temperatures, pitch and tar are almost unaffected, and the fibres of matting, canvas, &c., are acted on very gradually. Pieces of decaying wood or dried leaves must not be introduced with the water into the leaching vat, and if surface water is used it should always be carefully strained before being run in. A rough analysis of the water employed will often be serviceable, as it is frequently strongly alkaline in dry countries, and may be softened with advantage.

The absorption of chlorine by metallic oxides is the most frequent cause of waste, and, in the vat process, there are usually no efforts made to prevent this. Well roasted sesquioxide of iron (Fe_2O_3) is scarcely attacked by chlorine, especially if the temperature attained in the furnace has been high. If any magnetic oxide (Fe_3O_4), however, has been formed from over-heating or has been originally present in the ore, the absorption of chlorine is considerably greater, ferric chloride being formed and dissolved. Protoxide of iron is instantly converted into a mixture of chloride and sesquioxide of iron. Hydrochloric acid acts more rapidly than chlorine on all these oxides, but is nevertheless very slow in dissolving the well roasted sesquioxide. Metallic iron, which is sometimes accidentally introduced, is dissolved at once by both chlorine and HCl. The oxides of copper and zinc are quickly dissolved by chlorine, and still more readily by HCl. Lime and magnesia also readily absorb chlorine, forming hypochlorites, chlorates and chlorides, but hypochlorites are decomposed by any acid which may be present.

If any appreciable quantity of oxides capable of absorbing chlorine are present, it is cheaper to dissolve them by adding dilute sulphuric acid to the ore, and then, if possible, to leach out the soluble sulphates formed, before subjecting the ore to the action of the gas.

Amount of Chlorine required.—The amount of chlorine required varies greatly, both with the nature of the ore and the manner in which it is roasted. In order to roast pyrites dead, a long time in the furnace terminating at a high temperature is necessary, and the addition of salt may be desirable in order to chloridise oxides which would otherwise absorb the more expensive chlorine in the impregnation vat. These conditions in the furnace, however, may cause enormous losses by volatilisation, the endeavour to save a few pounds of chlorine in the vat causing the loss of 30 or 40 per cent. of the gold in the furnace. In ores where the percentage of copper, &c., is not large, and where, in consequence, salt need not be used in the furnace, the roasting may be finished at a high temperature without any disadvantage, and the consumption of chlorine may be thus reduced to a very low point. Butters states * that at his mill in Kennel, California, where all descriptions of concentrates and pyrites were treated by the Plattner process, the average consumption of chlorine was 12 lbs. per 2,000 lbs. of ore.

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* Ore roasted "dead" or "sweet" usually contains from 0·15 to 0·4 per cent. of sulphur chiefly in the form of sulphates, or locked up in uncrushed ore. The lower the amount of sulphur is reduced the better is the extraction of gold and the smaller the amount of chlorine required.

Leaching the Charge.—When it is judged that the impregnation has lasted long enough for all the gold to be dissolved, the excess of chlorine gas is removed, the lid is taken off, and water is added to the charge to wash out the soluble chloride of gold. The water may be added from below, and is then either allowed to overflow at the top, or is subsequently drawn off again at the bottom, the inflow being suspended. It is far more usual, however, to pour on water at the top, and let it flow out at the bottom.

The water must be added carefully, as otherwise the ore may pack unevenly, and channels may be formed through the mass, and the leaching thus rendered imperfect. Water is usually run from a tap on to a layer of gunny-sacking placed over the ore, by which it is distributed in a fairly even manner. It has been proposed to attach a coil of lead pipes, pierced with small holes, underneath the cover, and so sprinkle the water all over the ore in fine jets. In any case, water is added until it forms a layer 2 or 3 inches deep above the surface of the ore, and it is then allowed to stand until gas bubbles have ceased to rise through it, which happens in about half an hour. The stopcock below the false bottom is then opened, and the yellow- or blue-coloured solution (coloured by salts of iron, gold, and copper), which should have a strong odour of chlorine, is run slowly through a filter, consisting of a canvas bag, into a small barrel about 18 inches in diameter and 2 feet deep, the overflow of which passes by means of a launder or by rubber hose, to the precipitating tanks. Some of the ore and sand, escaping with the solution, is deposited in the canvas bag and barrel, but, if much slimes are present in the charge, either the canvas bag becomes clogged or the solution still remains turbid when it enters the precipitating vats. Water is supplied on the top of the ore as fast as it is drained away below, care being taken not to let the surface of the ore emerge from the liquid. The leaching is continued as long as any trace of gold can be detected in the issuing liquid by protosulphate of iron. As has been explained elsewhere (p. 26), a reaction is visible in clear solutions so long as the liquid contains more than two-thirds of a pennyweight of gold per ton of water, or one part of gold in one million of water. The strongly-coloured turbid solutions usually encountered in mills, however, are not capable of yielding distinct reactions, unless they contain much larger amounts of gold than this. In particular, when large quantities of copper salts are present in the solution, their strong bluish tints mask the slight discoloration due to a precipitate of a small quantity of metallic gold, and, moreover, they appear to interfere with the precipitation itself, in some cases at least preventing it from taking place.* It is always advisable to filter the solution by asbestos or filter paper before testing the liquid. Filter paper may be used, since it is very slow in precipitating gold from dilute solutions, even if they are neutral, and this action is completely stopped if free chlorine is present. It is better to test the clear liquid with stannous chloride under the conditions given at p. 27, since the presence of salts of copper does not seem to interfere with the reaction in this case, and, moreover, the amount of gold present can be determined in very dilute solutions with much greater accuracy than if ferrous sulphate is used.

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* *Eng. and Mng. Journ.*, Dec. 20, 1890.

Since a few minutes longer time occupied in leaching is of small moment, while the extraction of a few more grains of soluble gold from a ton of ore may be of the utmost importance in the long run, it is advisable to continue to leach at any rate until the water contains less than 1 part of gold in 5,000,000 (about 3 grains per ton), a point which can easily be determined by means of stannous chloride properly manipulated. The last charges of wash-water should not be mixed with the strong solution, but stored in other vats and used again for the first washings of other charges. In this way the amount of wash-water does not become excessive, although the tailings are cleaned effectually. Re-precipitation of dissolved gold in storage vats or impregnation vats is not to be feared, so long as there is an excess of chlorine present in the liquid, and this can easily be ensured by adding a small quantity to any solutions not smelling strongly of the gas.

The amount of water used varies according to the richness of the ore and the method of leaching adopted. It is usually about 2 tons of water to 1 ton of ore, but in most cases part of the water is used again in the next charge.

Precipitation of the Gold.—The precipitating vat is of the same materials as the leaching tubs, and may be from 5 to 7 feet in diameter and 3 feet deep. There is no false bottom, and the vat is often made wider at the bottom than at the top to prevent any adherence of the gold to the sides. The wood is protected by a coating of pitch or paraffin-paint, or is left without paint of any kind. The vat receives a smooth finish inside to facilitate perfect cleaning, and is set perfectly level to avoid loss of gold while the waste liquor is being drawn off. The precipitating solution of protosulphate of iron (the reagent which has been chiefly used in practice in the vat process) is usually introduced into the precipitating vat at the beginning of the filtering operations. Care must be taken not to introduce a wasteful quantity, as it is better to make up for any deficiency when the gold-solution has all been run in. This operation is conducted in such a way as to impart a circular motion to the contents of the vat, so that the solutions are mixed without hand-stirring, but the latter is often resorted to in addition, in order to make the precipitate settle better; flat wooden staves with round handles are used for the purpose. The contents of the vat may be tested with solutions of gold chloride and of ferrous sulphate to determine whether the precipitation is complete, and the precipitant present in excess.

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* C. Butters, *Eng. and Mng. Journ.*, Dec. 20, 1890.

If lead or lime is present, dissolved in the solution, it will be precipitated as an insoluble sulphate on the addition of the ferrous sulphate, and thus render the gold-precipitate impure and less easy to treat. The amount of lead in the solution is usually small, unless hot water has been used for leaching, and most of the lead chloride is, in any case, separated by the canvas filter. The usual method of removing the lime is to add sulphuric acid to the gold solution, and to let it stand for a few hours, when calcium sulphate crystallises out, forming a crust on the sides and bottom of the vat. The liquid is then drawn off and transferred to another vat for the precipitation of the gold. Instead of this method, Nelson A. Ferry, E.M., recommends* the addition of molasses to the leach, before the addition of the sulphate of iron. He dissolves 1 gallon of molasses in 30 or 40 gallons of water, and keeps it for use, determining the quantity to be added by a laboratory experiment. He states that in this way the precipitation of the lime is prevented, but a large excess of the ferrous sulphate should be avoided, and the liquid kept slightly acid. The gold then sometimes comes down flocculent at first, but soon changes to its normal condition.

The ferrous sulphate is usually prepared on the mill by dissolving iron in sulphuric acid. When precipitation is complete the liquid is allowed to remain at rest for some time, in order to allow the gold to settle to the bottom. The old practice was to leave it "overnight," but the length of time allowed has of late years been greatly extended. Butters* states that forty-eight hours is usually sufficient, but that sixty hours is better, and the determination of the extent to which the settling has progressed may be made by tapping the solution at various heights and filtering the liquid thus obtained. When a quart of liquid, drawn from a point 2 inches above the bottom of the vat, gives only a slight dark stain to a No. 7 Swedish filter paper, on being passed through it, the settling may be regarded as complete. C. H. Aaron quotes instances† where, after forty-eight hours settling, as much gold remained in suspension in the liquid which was drawn off as was equivalent to 50 cents per ton of the ore treated. The conclusion may be drawn from these statements that a certain amount of gold is inevitably lost by being carried away in suspension, but with care and patience the loss may be reduced to a low percentage, and even at the present day, in spite of the introduction of many other precipitants, ferrous sulphate is probably as widely used as it has been at any time in the past.

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* *Eng. and Mng. Journ.*, Nov. 28, 1885.

When the waste liquid has been drawn off by a floating siphon, more ferrous sulphate and fresh solutions from the leaching vat are poured into the vat, and the process repeated until enough gold has accumulated at the bottom to warrant a clean-up. This may take place at intervals of from a fortnight to three months. The clear liquid is drawn off as closely as possible, and the slime scooped out and filtered through paper, or, by means of a press, through canvas. Finally, the vat is thoroughly cleaned out by rinsing it with water which is run off through a plug-hole, level with the bottom, into a wash-tub. The gold precipitate is then dried carefully and fused in graphite pots, with salt, sand, nitre, borax, &c., as fluxes, according to the requirements of the case. If the precipitate contains any considerable amount of impurities (such as oxides and basic salts of iron), which is usually the case, it may be treated with hydrochloric acid before fusion. The bullion produced varies from 920 to 990 fine, the alloying metals consisting chiefly of iron and lead.

Cost of Working.—The cost of treating concentrates or ores by the Plattner process depends chiefly on the cost of roasting. In 1867, the total cost in California was stated by Küstel to be $14.55 per ton, but in 1872 it had been reduced to $11, the expense of roasting being in each case about two-thirds of the whole. At the works of the Plymouth Consolidated Mining Company, California, in 1886,‡ the cost of treating 100 tons per month was $9.40 per ton (the roasting accounting for $4.60 per ton, or nearly one half), and at the Providence Mine, in the same

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* *Eng. and Mng. Journ.*, Dec. 20, 1890.

† *Eighth Report Cal. State Mineralogist*, 1888, p. 836.

‡ *Trans. Am. Inst. Mng. Eng.*, 1886.

State, it was about $6.30, without including the expenses of general supervision, interest on first cost, and depreciation of plant.

It was estimated in 1888* that, generally, throughout California, a plant, capable of treating 6 tons of concentrates daily, cost from $6,000 to $7,000 for its erection, while the cost of extraction was about $10 per ton, and the proportion of the gold extracted was from 90 to 92 per cent. It has, however, been more recently stated by G. F. Deetken that the cost of vat chlorination under favourable conditions need not exceed from $3 to $4 per ton, but he gives no details or facts in support of his statement, and it seems probable that the conditions required are too favourable to be expected.

Use of Liquid Chlorine.—At the Utica Mine, California, the Plattner process is in use.† The concentrates are roasted in long reverberatory furnaces, allowed to cool, moistened with sulphuric acid and water and sieved into wooden vats. Chlorine gas derived from a drum of liquid chlorine is then introduced into the charge, and after forty-eight hours the ore is leached in the usual way. The cost of treatment was formerly $7.80 per ton, when the chlorine was generated from manganese dioxide, salt and sulphuric acid, and only $6.90 per ton after the introduction of liquid chlorine. One drum of liquid chlorine weighs 300 lbs., holds 115 lbs. of chlorine, and takes the place of 972 lbs. of manganese dioxide, 1,080 lbs. of salt, and 2,160 lbs. of sulphuric acid. The drum is 10 inches in diameter and 5 feet long, and costs $40 at the Utica Mine. The time of introducing the gas into a charge is reduced by its use from six or eight hours to twenty minutes.

The Plattner Process at the Plymouth Consolidated Gold Mining Company, Amador County, California.‡—The ore contains 11 dwts. of gold, mostly in the free state. It is crushed in a stamp battery, No. 8 screens (40 mesh) being used, passed over a length of 20 feet of amalgamated plates (the upper one of which is copper and the rest silver-plated) and then concentrated on Frue vanners. The concentrates amount to from 1¼ to 1½ per cent. of the weight of the ore, and contain from 5 to 10 ozs. of gold per ton. They are treated at the rate of 100 tons per month, being kept damp until charged into the roasting furnace, to prevent the formation of lumps. A "Fortschaufelungsofen" is used for roasting, 80 feet long by 12 feet wide, its hearth consisting of a long continuous plane, holding three charges at one time, which are kept separate. The three stages are called the "drying," "burning," and "cooking" stages. In the

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* *Eighth Report Cal. State Min.*, 1888.

† *Eng. and Mng. Journ.*, April, 22, 1899, p. 467.

‡ For a more complete description, see that given in the *Eighth Report Cal. Stat. Min.*, 1888, of which the account appended is an abstract.

(middle) burning stage, the bed of ore is kept thin and occupies double the space of each of the other charges. The furnace is worked by three eight-hour shifts of one man. The charges weigh 2,400 lbs., including 10 per cent. of moisture, and on an average contain 20 per cent. of sulphur. Just before the sulphur ceases to flame, $\frac{3}{4}$ per cent.—*i.e.*, 18 lbs.—of salt is added.

The chlorinating vat is 9 feet in diameter and 3 feet deep: it holds 4 tons of ore. The filter-bed is 6 inches deep, and consists of, (*a*) at the bottom, wooden strips, $\frac{3}{4}$ inch wide, placed 1 inch apart; (*b*) above this, 6-inch boards placed 1 inch apart and laid across the strips; (*c*) coarse lumps of quartz, diminishing upwards to fine stuff; and (*d*) at the top, a cover of 6-inch boards placed similarly to the lower layer, but crosswise to them. Chlorine is introduced on both sides of the vat, which is left luted-up for two days, and then leached for four or five hours, the tank being kept full of water during the operation. A gunny-sack protects the surface of the ore from the direct impact of the water from the hose. The ore in the impregnation vat contains about 6 per cent. of water (crumbling after it has been sieved), and is sieved into the vat through a screen of $\frac{1}{2}$-inch mesh.

The gold solution is acidulated with sulphuric acid to precipitate the lead, and ferrous sulphate is then added to it in another tank, after which it is allowed to settle for two days before the supernatant liquor is siphoned off. The gold is left to accumulate for fourteen days and the wet gold is then filtered and fused in graphite pots. The average extraction is from 95 to 96 per cent. of the gold, and 7 dwts. per ton are left in the tailings. All wood is protected from the action of the acids by paraffin-paint. The cost of milling is said to be 39 cents per ton of ore, and the cost of concentration, roasting, and chlorination, $13.40 per ton of concentrates.

The Plattner Process at the Alaska Treadwell Mine.—These works turned out more gold than any other chlorination mill in the United States, until the Golden Reward and other Dakota mills were started. The old-fashioned vat process in use has been adopted after thousands of dollars have been spent in testing newer methods. The works have been in active operation since the year 1884. The material treated consists of the sulphides collected by Frue vanners from a stamp battery, and contains 40 per cent. of sulphur, mostly as iron pyrites, although the percentage of copper is increasing. The gangue is quartzose, containing from 2 to 5 per cent. of calcite, which necessitates the addition of salt in the roasting furnace.

The roasting was first effected in Brückner cylinders, which were abandoned owing to the large amount of fuel consumed and the enormous losses by dusting and volatilisation (which are said to have amounted to 30 per cent. of the gold). The automatic Spence furnace was then tried and proved to be useless until it was used as a reverberatory. Six were erected, and the cost of roasting was reduced by one-half, but the capacity was small (10 tons per day in the six furnaces) and the consumption of fuel great, and a reverberatory furnace being erected, was found to be more satisfactory. The Spence furnaces were accordingly discarded, and 25 tons of concentrates were roasted per day in five reverberatory furnaces of 13 by 65 feet, inside measurement, at a cost of $3½ per ton. The ore is not roasted quite dead, Mr. Burfeind, the superintendent, having decided that when salt is added the best results, with least volatilisation, are obtained by leaving some sulphates undecomposed. He gives it as his opinion that improper roasting is responsible for all losses of gold in the tailings and by volatilisation.

The roasted ore is spread on the cooling floor, wetted down and sifted carefully into vats, each of which holds 4½ tons, and is then impregnated with the gas. This operation occupies four hours, after which the vat is left untouched for thirty hours, fresh gas being forced in a few hours before leaching. The leaching usually requires twelve hours. The tailings are sampled and assayed, and, if found sufficiently poor, are sluiced into the sea.

The solution is run into collecting tanks and thence to the precipitating vats, which already contain the necessary amount of ferrous sulphate in solution. The precipitation is complete when all the solution has been run in, or when the vat is full. It is then stirred briskly for a few minutes and left to settle for from eighteen to twenty-four hours, when the supernatant liquor is siphoned off and passed through a large filter. The supernatant liquor usually contains from 23 to 25 cents of gold per ton of material treated, and this is all saved in the filter.

The clean-up is made twice a month, the drying and melting of about 750 ozs. of gold being done by one man in one day. The gold is washed into a small tub, allowed to settle, and the supernatant liquor returned to the precipitating vat. The gold is dried in an iron pan without filtering, and melted with a little borax.

Each one of the chlorination vats, holding 4½ tons of ore, cost $50, and lasts for three years without any repairs. The filter in it costs only the price of a few gunny-sacks, and lasts for six months without any attention. The other vats are expected to last a lifetime. The ferrous sulphate is prepared on the works from sulphuric acid and scrap-iron. In 1890, the cost was at the rate of about $10 per ton, the yield in gold being over $40 per ton. The results for the period June, 1890, to May, 1892, were as follows:—

Year.	Tons Chlorinated.	Yield per Ton.	Cost per Ton.		
			Labour.	Supplies.	Total.
June, 1890, to May, 1891,	5,869	2·02 ozs.	$5.03	$3.99	$9.02
June, 1891, to May, 1892,	6,177	1·52 ,,	4.08	2.81	7.61

CHAPTER XIV.

CHLORINATION: THE BARREL PROCESS.

Historical.—The use of revolving barrels for chlorinating ores was perhaps suggested by the old Freiberg method of barrel amalgamation. It has already been mentioned, p. 239, that Dr. Duflos used a revolving barrel in some of his experiments at Breslau, in 1848, and obtained results almost identical with those given by the vat percolation method. He, therefore, preferred the latter as being cheaper. The next mention of revolving barrels seems to have been contained in a patent taken out by Mr. De Lacy in Victoria, in 1864. The process thus patented appears to have been tried there, probably on a small scale, but certainly never passed into general use and was soon forgotten. In 1877, Dr. Howell Mears, of Philadelphia, patented a process which had some points of resemblance to that described by De Lacy. This process was gradually improved in practice, after having been adopted by several mines in the United States, and in particular the improvements introduced by Adolph Thies, in 1881, caused the name of the Thies process to be applied to the amended method of procedure. In 1887 the barrel process was re-introduced with some modifications into Australia by Prof. J. Cosmo Newbery and Mr. C. J. T. Vautin, who applied it to the ore of the Mount Morgan Mine, where it was successfully worked for some years before it was replaced by the vat solution process. In 1889 and succeeding years the barrel process was developed in the United States, and appeared likely to become of great importance. Of late years, however, it has been generally regarded with less favour than the cyanide process.

The Mears Process.—In this process, the roasted ore was charged into lead-lined iron cylindrical barrels, together with enough water to make an easily flowing pulp; chlorine was then forced in under pressure through the hollow trunnion of the barrel, which was revolved until the gold had been dissolved. The pressure of chlorine was stated to be as much as 40 or 50 lbs. to the square inch. The manhole of the barrel was then opened and the ore discharged by gravity into a leaching vat below, where the soluble gold was washed out and precipitated by any of the known methods.

The process was formerly in use at the Phœnix and Haile Mines in Carolina, and at the Bunker Hill Mine, California.

The Thies Process.—In this process the chlorine gas was generated inside the barrel itself by means of bleaching powder and sulphuric acid. The method was simplified in this way, a a number of leaky joints dispensed with. Thies found that moderate pressure of chlorine, of a few pounds to the squa inch, was enough. His barrel is shown in Fig. 64. It w about 5 feet long, and held from 2,000 to 2,500 lbs. of ore.

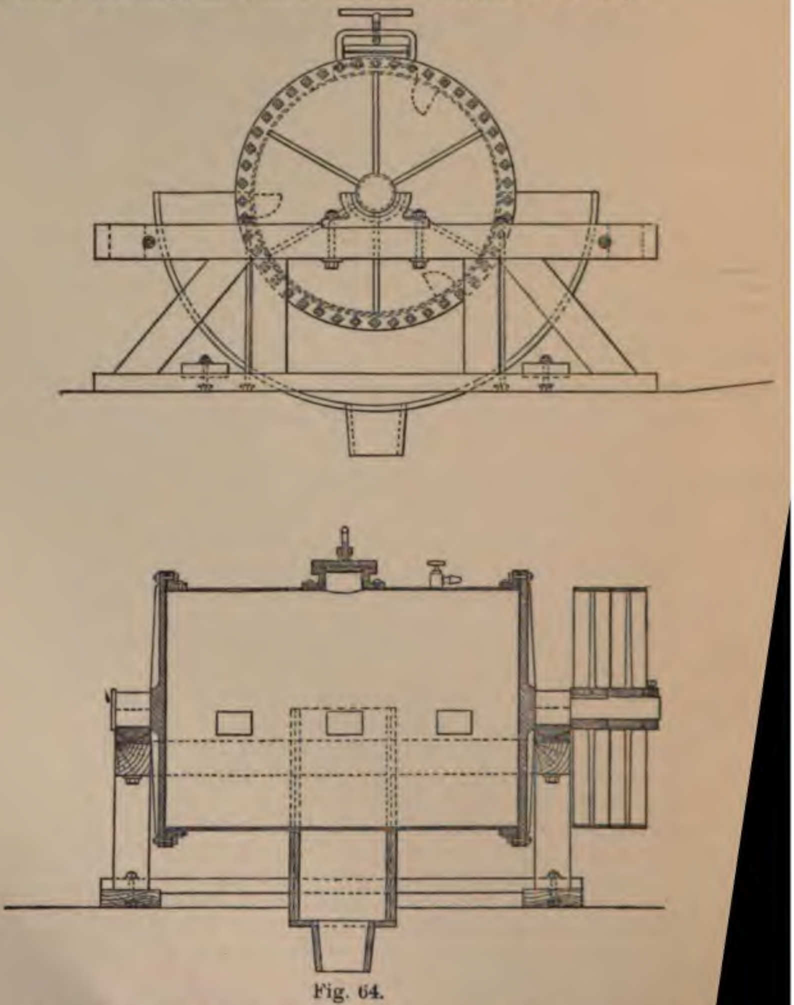

Fig. 64.

The Thies Process at Deloro.—At Deloro the ore consisted o arsenical pyrites with a gangue of quartz and calcite. It was crushed by rolls, concentrated, and the concentrates roasted i

White-Howell cylinders. The chlorinating barrels were in two sizes, the smaller one being 44 inches long and 40 inches in diameter inside. The capacity of this was 32½ cubic feet, and the charge was 1 ton of ore, 120 gallons of water, 36 to 54 lbs. of bleaching powder (= 12 to 18 lbs. of chlorine), and 42 to 63 lbs. of sulphuric acid. The barrel was revolved for 1½ to 3 hours, and the charge was then discharged into a filter-vat, and washed with the aid of an exhaust-pump to quicken the leaching. The clear solution was precipitated by sulphuretted hydrogen, which was mixed with air before being forced to the bottom of the solution.

*The Thies Process at the Phœnix Mine, Concord, North Carolina, and the Haile Gold Mine, South Carolina.**—Chlorine was at first forced in through a gooseneck, passing through the trunnion (Mears' process), but it was found that the gooseneck wore out quickly, and that its inner lead lining collapsed when the gas was exhausted. It was, therefore, discarded after two years' experience, and the chlorine made inside the barrel by means of sulphuric acid and bleaching powder. The chlorinator was 42 inches in diameter and 60 inches long, the capacity being 48 cubic feet, and the charging-hole 6 inches in diameter. The lining consisted of 10-lb. to 12-lb. sheet-lead. The weight of the charge was from 2,000 to 2,500 lbs., to which from 100 to 125 gallons of water were added—*i.e.*, enough to make an easily flowing pulp. At the Phœnix Mine, where the ore contains copper, one-half of the acid and bleaching lime was added first and the barrel turned at the rate of 15 revolutions per minute for three to four hours. For the Phœnix ores, this half charge of chemicals consisted of 20 lbs. of lime and 25 lbs. of acid. After this time had elapsed the barrel was opened, and the other half of the chemicals added, after which the barrel was rotated for from two to three hours longer. The charge was then let fall on to a filter-bed, 6 feet long and 8 feet wide, the depth of the pulp on it being 4 inches, while the filter-bed was 5 inches thick. In all, 2 tons of water were used to leach 1 ton of ore.

The bottom of the filter-bed consisted of perforated glazed tiles, and stones, gravel, and sand were successively piled on these as usual. The gold was precipitated by ferrous sulphate, and an excess of sulphuric acid was kept in the charge to avoid the precipitation of calcium sulphate with the gold. The large amount of lime and acid added to this ore was necessitated by the presence of copper. The solutions were allowed to settle for three days and then run through tanks filled with scrap-iron, by which the copper was precipitated.

At the Haile Mine, where iron pyrites, free from copper, was treated, only 10 lbs. of lime and 15 lbs. of acid per ton of ore were used. Ten tons of roasted ore were treated in the large barrels per day of ten hours, and 94 per cent. of the gold extracted.

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* *Eighth Report Cal. State Min.*, 1888, p. 836.

The cost at the Haile Mine was $4.65 per ton, without reckoning the expenses of superintendence and assaying, the cost of roasting being $2.70, and of chlorination, &c., $1.95. The following details are given on the authority of Mr. Adolph Thies, the inventor of the process used and the manager of the mill :—

Fire-assay value of ore, delivered to the stamps, $4.50, of which $1.45, in free gold, was caught on the plates.

Average assay value of raw concentrates,	$30 per ton.
Sulphur in raw concentrates,	40 to 45 per cent.
Value of roasted concentrates,	$50 per ton.
Average value of tailings from chlorination,	$2 per ton.

COST OF ROASTING.

½ Cord of wood at $1.40,	$0.70
Two labourers at $1,00,	2.00
Per ton of roasted ore,	$2.70

COST OF CHLORINATING.

10 lbs. chloride of lime at 3 cents,	$0.30
15 lbs. commercial sulphuric acid at 2 cents,	.30
Two labourers, ¼ day at 90 cents,	.45
Chlorinator, ¼ day at $2.00,	.50
Power,	.12½
Sulphuric acid for ferrous sulphate,	.12½
Wear and tear,	.10
Superintendence,	.05
Per ton of roasted ore,	$1.95

TOTAL COST.

Per ton of roasted concentrates,	$4.65
Per ton of raw concentrates,	3.50

A lead-lined barrel which had been in use for five years at this mill showed no signs of wear, the abrasion of the lining by the ore being evidently almost *nil*.

General Principles in Barrel Chlorination.—The barrel is revolved at the rate of six to ten revolutions per minute by a belt and pulley (the latter being fixed on a continuation of the trunnion), or by means of cog-wheels, or by a friction clutch. The ore is kept constantly stirred and tumbled about by the revolution of the barrel, the advantages gained from this course being as follows :—

1. Every particle of ore is exposed equally to the action of the chlorine.

2. There can be no local exhaustion of the solvent near coarse particles of gold.

3. It was formerly argued that the gold would be cleaned by attrition, and in particular that silver chloride, formed by

the action of chlorine, would be rubbed off the surface of particles of native gold-silver alloys, and clean surfaces of gold would thus be continually offered to the attack of the chlorine.

Hofman and Magnuson,* however, have shown that there is a limit to this action, if it exists. They found that with pure gold and quartz grains an excess of chlorine in a rotating bottle dissolved pure gold a little faster than gold containing 10 per cent. of silver. The alloy gold 80, silver 20 was almost as readily dissolved by a supersaturated solution of chlorine as pure gold, but the alloy gold 70, silver 30 was acted on much less rapidly. The solvent power of somewhat weaker solutions was far more seriously interfered with by the presence of 20 per cent. of silver in the alloy.

In charging-in, the water is added first, being run in by a hose through the manhole, until it reaches a certain mark on the inside of the barrel. The amount of water to be used varies with the nature of the ore—roasted pyrites absorbing much more water than siliceous ores; the quantity required is usually from 40 to 60 per cent. of the weight of the ore—*i.e.*, from 80 to 120 gallons of water per 2,000 lbs. of ore. This is enough to make an easily flowing pulp. It is much more than that employed in the Plattner process, where it is necessary to keep the mass porous, so as to enable the gas to pass through it. In the barrel, however, this reason for limiting the amount of water does not exist, and the ore is really treated by a solution of chlorine water. If too small a quantity of water is used, so that the pulp is not quite free-flowing, lumps are formed which are not broken by the revolution of the barrel, and these lumps are not perfectly chlorinated.

The ore is let fall into the barrel down a shoot from an overhead hopper, which may conveniently be made to contain the exact quantity of ore required for a barrel charge. The ore should be perfectly dry (not cooled after roasting by too much "wetting down"), as otherwise it sticks in the hopper, instead of sliding freely down the shoot. The latter may be conveniently made of canvas, so that it can be looped-up out of the way when not in use. The chemicals (bleaching powder and sulphuric acid) may be added in one of two ways. Either the lime is thrown into the water before the ore is added, and the acid subsequently poured upon the upper dry surface of the latter, just before the manhole is closed; or the acid is poured into the water, through which it sinks to the bottom without mixing, the ore let fall next, and the lime added on the top. This latter system is perhaps the better, as, if it is used, the generation of chlorine is never begun before the barrel commences to revolve, provided ordinary care is taken.

The amounts of bleaching powder and sulphuric acid to be added depend on the nature of the ore, and the care with which it has been roasted. If there is nothing in the ore which is open to attack except gold, the amount of chlorine actually absorbed in the course of the chemical reactions is extremely small, 1 oz. of gold requiring only 0·54 oz. of chlorine to convert it into the soluble trichloride.

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* *Bulletin Am. Inst. Mng. Eng.*, March, 1905, p. 421.

The bleaching powder used should be of the finest quality obtainable, to prevent the introduction of an unnecessarily large amount of sulphate of lime into the charge. Bleaching powder usually has assigned to it the formula $Ca(OCl)Cl$, or $Ca(OCl)_2 + CaCl_2$, and may be, as the latter formula would indicate, a mixture of hypochlorite and chloride of lime. The reaction with acid is usually expressed thus—

$$CaCl_2 + Ca(OCl)_2 + 2H_2SO_4 = 2CaSO_4 + 2H_2O + 2Cl_2.$$

This equation certainly does not accurately represent what happens, as much less chlorine is liberated than is indicated by it. The amount of "available" chlorine (*i.e.*, that which is liberated by the action of acids) contained in commercial bleaching powder varies from 20 to 35 per cent. Bleaching powder is gradually decomposed by the carbonic anhydride in atmospheric air (chlorine being liberated), and must consequently be kept separated from it as completely as possible. Even if preserved in hermetically sealed vessels, however, it suffers a slow change, by which some chlorate of lime is formed and the amount of available chlorine reduced. Under these circumstances, it is necessary to re-determine the value of the bleaching powder at short intervals of a few days. The shortest and best method of effecting this is to grind a sample in a stoneware mortar under water, and add the emulsion to an excess of a solution of potassium iodide. The whole of the available chlorine instantly displaces an equivalent quantity of iodine, which is set free and may be readily estimated by a standard solution of hyposulphite (thiosulphate) of soda in the usual manner. From this the amount of available chlorine in the sample is calculated. The whole operation can be performed in from ten to fifteen minutes when the standard solution has been prepared.

The amount of chlorine added to the barrel must be enough to yield a considerable excess at the end of the operation. It is usually considered that this excess should be enough to make the solution smell strongly of the gas when exposed to the air at the ordinary temperature and pressure. This is nearly equivalent to stating that at the end a saturated solution of chlorine in water should remain. If the water used is 100 gallons per ton of ore, the minimum amount of chlorine to be used should be, on this supposition, from 6 to 8 lbs. per ton of ore. This quantity may be taken as one well adapted for the efficient chlorination of ordinary ores. Nevertheless, much smaller quantities have been used with complete success on certain ores, and, as has been already stated, no general rule, applicable to all cases, can be laid down. Thus certain ores from Dakota, containing only 1 or 2 per cent. of sulphur, and consisting chiefly of silica, were chlorinated by the author in revolving barrels, using only 3½ pounds of chlorine per 2,000 pounds of ore. Even in this case, there was a strong excess of chlorine in the ore after the solution was complete, and the amount used could probably have been still further reduced without lowering the percentage extraction of gold. This ore contained 10 dwts. of gold to the ton, and over 80 per cent. was extracted. Most of the gold in the residues was locked up in coarse particles of ore. This was an extreme case, and it is seldom that so little chlorine is sufficient. When much oxide of copper or other metallic oxides capable of absorbing chlorine are present, or if the ore is insufficiently roasted, the pressure rapidly falls off, and a further addition of chemicals may become necessary, as at the Phœnix Mine. The total pressure exercised by the air and chlorine is sometimes made equal to two atmospheres—*i.e.*, one atmosphere in excess of the normal—and this may be taken as the greatest pressure which it is advisable to maintain inside a barrel. If greater pressures are used, either very expensive barrels are required, or else the valves, manhole, &c., soon begin to leak.

It must not be forgotten that, at higher temperatures, water dissolves less chlorine than the amount mentioned in the last paragraph, and, consquently, a pressure of chlorine equal to that of the atmosphere will be obtained with the use of smaller quantities of chemicals. Thus at 40° C., with the other conditions identical with those given above, the minimum amount of chlorine required to yield a saturated solution at the end of treatment is only about 4·3 lbs., and, as the temperature rises above this, the quantity of chlorine necessary falls off rapidly.*

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* For the amount of chlorine dissolved by water at different temperatures, *vide* Schönfeld, *Ann. Chem. Pharm.*, vol. xciii., p. 25; vol. xcvi., p. 8. The results are quoted in Roscoe & Schorlemmer's *Treatise on Chemistry*, vol. i., p. 123, 1881. The following results, giving the amounts of chlorine contained in saturated solutions, have been abstracted, the volumes of chlorine being at 0° and 760 mm. :—

Temperature.	Volumes of Chlorine Gas in One Volume of Water.	Weight of Chlorine in One Litre of Water. Grms.	Temperature.	Volumes of Chlorine Gas in One Volume of Water.	Weight of Chlorine in One Litre of Water. Grms.
10°	2·585	8·203	30°	1·750	5·554
15°	2·368	7·515	35°	1·555	4·935
20°	2·156	6·842	40°	1·365	4·332
25°	1·950	6·174			

The temperature of the barrel is raised by several degrees by the chemical action. The exact amount in practice has not been determined.

The relative amounts of bleaching powder and acid used is varied with different ores. Theoretically, according to the equation given above, 7 parts of chloride of lime require 6 parts of sulphuric acid for complete decomposition, but practically a little more sulphuric acid is always added, because it is desirable to maintain an excess of the acid in the charge. This is done in order to prevent lead and lime from getting into solution as chlorides, which would entail a loss of chlorine, and also to assist in dissolving any oxide of copper that may be present, which otherwise would also absorb chlorine. The proportions added are usually 1 part of bleaching powder to 1½ or 2 parts of sulphuric acid of 66° B.

The time occupied in chlorinating usually varies from three to six hours. The continued presence of an excess of chlorine gas should be tested from time to time by opening a small valve momentarily. If the presence of the free gas cannot be detected, the barrel must be opened and further supplies of chemicals added. The amount of pressure in the barrel is not alone sufficient to prove the presence or absence of an excess of chlorine, as other gases may be generated and exert considerable pressure. When the chlorination is finished, the excess of chlorine is discharged by a hose-pipe outside the building, the barrel is filled up with water, again revolved, and the liquid decanted on to large, shallow filter-beds. The barrel is again filled up, revolved, and decanted as before, and, finally, the whole charge is emptied out and another wash-water given to the charge on the filter. In comparing this method of washing by decantation with that of direct filtration, usually adopted, A. Thies found that, with similar charges, the amount of water used in the latter case was nearly double, and the time occupied much longer, while the tailings contained 5½ dwts. of gold per ton against about 1 dwt. when washed by decantation. He also found that there is little difficulty in filtering through a bed of fine ore from 3 to 4½ inches thick, but if the thickness of the bed is greater, direct leaching becomes very tedious and ineffective, and decantation is much better.

The leaching vats are usually constructed of wood, which is either lined with lead or coated with tar and pitch. They are usually round, but at Bunker Hill they were rectangular, measuring 6 feet by 18 feet, and 18 inches deep. They are lined with lead, and incline towards the drain hole, where the bottom is 1 inch lower than at the other side of the tank. The filter-bed was made of quartz-pebbles, gravel, and fine sand.

Mechanical Difficulties of Leaching.—The difficulties of leaching vary enormously with the character of the ore and the treatment to which it has been subjected. Concentrates are among the best leaching ores, as, even if they were originally in a state of extremely fine division, the oxidising roasting in many cases appears to cause an agglomeration of the particles into porous granules, which do not pack down readily, and do not resist the passage of liquids. A small quantity of red oxide of iron in a very fine state of division is often present in roasted concentrates, and this is carried away by the water, passing into and partly through the filter-bed and even appearing in the precipitating vat. This material, on settling to the bottom of the gold solution, often appears to carry down with it some of the gold, forming a layer of rich slimes. In some cases, these iron oxide slimes are present in roasted pyrites in such quantites that leaching is greatly interfered with, and made very tedious. Siliceous ores, if properly pulverised usually present no difficulty in leaching, but aluminous ores are exceedingly troublesome. At Mount Morgan, Queensland, the ore formerly consisted chiefly of hydrated oxides of iron, which offered the greatest possible resistance to leaching if treated raw, but, if roasted, the loss of their water of hydration was found to be accompanied by a remarkable agglomeration of the particles of ore, and ordinary gravitation leaching was thus rendered possible.

In order to quicken the process of leaching various appliances have been suggested. Different forms of vacuum pumps have been used, the amount of air in the space below the filter-bed being reduced by them, and the liquid thus forced through by atmospheric pressure. In 1889, at the Colorado Gold and Silver Extraction Company's mill at Denver, the effect of the use of increased pressure of air applied directly on the surface of the liquid was tried. A special cast-iron vat was constructed capable of sustaining an internal pressure of 100 lbs. per square inch, and furnished with valves, so that air and water could be simultaneously pumped into it. It was found that ores, which entirely prevented the passage of water through them, even after a vacuum of 20 inches of mercury had been established beneath the filter-bed, could be leached with great speed under a pressure of from 30 to 50 lbs. per square inch. Moreover, when the leaching was complete, the ore could be freed from the water more completely by the passage of a current of air through it than by gravitation alone. This method, which was suggested by D. Dennes, the Company's engineer, was modified by J. E. Rothwell, who placed the filter-bed inside the barrel. The chief objection to it seems to lie in the additional expense incurred in the construction of the barrels or leaching vats and in working the pumps.

Mr. Riotte long ago suggested* that the wash-water should be thoroughly mixed with the ore by agitation, and then removed as completely as possible by squeezing in a screw filter-press. He used a pressure of from 25 to 30 lbs. per square inch, and found that the average amount of moisture retained in the ore after being squeezed was about 6 per cent. As this would cause the retention in the tailings of from 3 to 6 per cent. of the soluble gold, according to the amount of wash-water used, the method is obviously inapplicable to rich ores, which would require to be subjected to treatment twice. Centrifugal leaching has also been proposed, and it is stated that the Mount Morgan ore can be leached in this way without previous roasting.

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* *Eng. and Mng. Journ.*, March 31, 1888.

The Precipitation of Gold.—A number of reagents have been used or proposed for this purpose. An account of the chief methods is given below.

Ferrous Sulphate.—This was used by Plattner and is still widely employed. The chief difficulties in connection with it are due to the slowness with which the precipitated gold settles and to the difficulty of collecting it by filtration. Some account of its use has already been given on p. 251. It is made by dissolving iron in sulphuric acid, with the aid of heat, and the crude solution thus prepared, which is in most cases added direct to the gold solution, always contains some free sulphuric acid. Precipitation takes place according to the equation—

$$2AuCl_3 + 6FeSO_4 = Au_2 + Fe_2Cl_6 + 2Fe_2(SO_4)_3$$

The oxidation of the ferrous salt is also effected in other ways, notably by the excess of free chlorine present in the solution, so that much more sulphate of iron is required than is indicated by the equation. The difficulty of collecting and saving the precipitated gold has already been dwelt on. The gold settles better if it is well stirred, and Aaron recommends an addition of more sulphuric acid and vigorous stirring, two hours after the precipitation is complete, as a means of assisting the settling. Besides gold, the only other metals precipitated by ferrous sulphate are those which form insoluble sulphates—viz., lead, calcium, strontium, and barium. The last two of these are rarely present, and the others are dealt with in the manner already described above. Basic iron salts are not precipitated if enough free sulphuric acid is present, and when precipitated, they may be removed from the gold by treatment with acids, or by slagging them off in the furnace.

Other proto-salts of iron are equally efficacious, and ferrous chloride has been used in the *aqua regia* parting process described in Chapter xviii.

Sulphuretted Hydrogen.—This was formerly made at Deloro by heating paraffin and sulphur together, and the gas diluted with air was forced through the solution by means of a small air-pump. The use of the air was to keep the solution agitated, and to expel part of the chlorine mechanically, and so economise the sulphuretted hydrogen, which is decomposed by chlorine. The equation for this decomposition is given by Langguth * as follows:—

$$H_2S + 4H_2O + 8Cl = H_2SO_4 + 8HCl$$

but it is more likely that the greater part of the change is represented by the well-known equation—

$$H_2S + Cl_2 = 2HCl + S$$

although a small amount of sulphuric acid may be formed at the same time. The precipitate of sulphur thus formed before and during the precipitation of the gold, which begins before the whole of the chlorine has been destroyed, is to be avoided, and Langguth has, therefore, suggested the use of sulphur dioxide, generated by burning sulphur to destroy the free chlorine, the reaction being as follows—

$$Cl_2 + SO_2 + 2H_2O = H_2SO_4 + 2HCl$$

When almost all the chlorine has been thus converted into hydrochloric acid, the point being determined by testing some of the liquid with a few bubbles of sulphuretted hydrogen, the passage of SO_2 is stopped, and sulphuretted hydrogen, now generated by the action of sulphuric acid on iron matte, is forced into the solution, destroying the last traces of chlorine and precipitating the gold. This system was introduced at the Golden Reward Chlorination Works in 1891, and was subsequently adopted at other mills in the United States, but the use of SO_2 was afterwards abandoned.

The gold is precipitated as a sulphide mixed with more or less sulphur, the reaction being represented by the following equation—

$$2AuCl_3 + 3H_2S = Au_2S_3 + 6HCl$$

It is said to take less than an hour at the Golden Reward Works to precipitate the gold from 5,000 gallons of solution (resulting from the lixiviation of from 25 to 50 tons of ore). The liquid is quite cold, but the precipitate is in a collected, voluminous and flocculent form which settles quickly. It is left undisturbed for two hours, and the liquid is then drawn off to within 4 inches of the bottom of the vat, and passed through a filter-press, provided with a set of heavy, canton-flannel filter-cloths. The head of liquid used for filtering is 25 feet, and the filtration is said to occupy from three to four hours, according to the amount of sulphides already contained in the filter. When the latter is full, a small air-pump is connected with it and a current of air passed through it for an hour to dry the mass of sulphides into hard cakes, which are easily handled and

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* *Eng. and Mng. Journ.*, Feb. 14, 1891.

removed. The precipitate is then roasted in iron trays, which are placed in cast-iron muffles and heated only from the top, no stirring being necessary, so that the loss from dusting is small. The filter-cloths are burnt with the precipitate, when they have become either clogged with sulphides, or untrustworthy owing to the action of the acid liquors. The precipitate is then melted down with a little borax and nitre, the total loss in handling being very small. The bullion is about 900 to 950 fine in gold, the remainder consisting chiefly of silver, copper, lead, and arsenic. The bulk of the precipitate remains at the bottom of the vat. It is allowed to accumulate for a fortnight, and is then filtered and treated as above. The slag resulting from the fusion of the gold is crushed and melted down with litharge, and the reduced lead cupelled (Rothwell).

It has usually been assumed that all the lead, copper, and silver contained in the liquids are precipitated with the gold, and that if much copper is present, the bullion will be very base, and Langguth suggested the removal of the copper from the precipitate by dilute nitric acid. Rothwell, however, has stated (*Mineral Industry for* 1896, p. 278) that, under normal conditions, the gold can be precipitated, forming a bullion from 820 to 960 fine, whilst the copper remains in solution together with barely a trace of gold.

Sulphurous Acid, which has been already mentioned as a cheap agent for destroying free chlorine, precipitates gold very completely from dilute solutions, but its action in the cold is slow until the liquid is almost saturated with the gas (water absorbs 39 volumes of the gas at 20°), and the gold settles slowly. At higher temperatures the action of the gas is much more rapid and satisfactory, and very little is wasted in saturating the liquid. As a precipitant, the gas does not appear to present any advantages over ferrous sulphate.

Charcoal.—In 1801, Henry referred to the reducing action of charcoal on chloride of gold in solution, and observed that the gold will be precipitated on the charcoal, if the solution is either exposed to the direct light of the sun or heated to 212° F.* In 1869, in Percy's laboratory, some sticks of wood-charcoal were immersed in water, and 32·50 grains of gold in the form of chloride were added on August 7. 85 more grains of gold, in the form of chloride, were added on November 3, 1869, and the bottle was left to stand. This bottle is in the Percy collection, at South Kensington, and the surface of the charcoal is now coated over with metallic gold which shows on its surface the fibres and vessels of the stem.

Vegetable charcoal was adopted as a precipitant at the Mount Morgan Mine, Queensland, in 1887. The method adopted there is as follows:—The solution is heated, the free chlorine being thus expelled; the liquid is then made to run slowly through large shallow tanks, each 10 feet by 11 feet and 4 feet 6 inches deep (see Fig. 65).* They are built of brick on a concrete foundation, and lined with Portland cement and tar. The layer of charcoal in them is 2 feet deep, rests on a filter-bottom, and is covered by thick perforated sheets of lead. The liquor flows downwards through three charcoal filters in succession, 660 cubic feet of charcoal being sufficient for the treatment of the liquor from about 100 tons per day of 11 dwts. ore. In preparing the charcoal for the filters,† it is crushed, and all that passes a 20-mesh and remains on a 30-mesh screen is called "coarse," whilst that which passes the 30-mesh and remains on a 40-mesh screen is called "fine." All that passes 40-mesh is thrown away. Two cubic feet of fine are used to 1 cubic foot of coarse, the coarse being at the top and bottom. The charcoal is well rammed in with the foot, especially round the walls. For seven months the liquor running away from these filters assayed in one mill 0·76 grain gold per ton, and in the other 1·89 grains per ton. After passing the charcoal, all liquors are run through a concrete tank with a bed of sawdust 1 foot thick and more gold caught in this way. When the charcoal is coated sufficiently with gold, it is burnt in small furnaces furnished with dust chambers. The ashes were formerly amalgamated, and the rich slimes sold to the smelters. Afterwards the ashes were fluxed in crucibles and eventually in a small reverberatory furnace, the fluxes used being soda ash and borax, which were mixed with the ashes and then moistened with water. D. Clark states that the charcoal ash contains from 20 to 40 per cent. of gold.‡ The slag contained from 20 to 30 ozs. per ton. Animal charcoal cannot be used owing to the difficulty of burning it afterwards.

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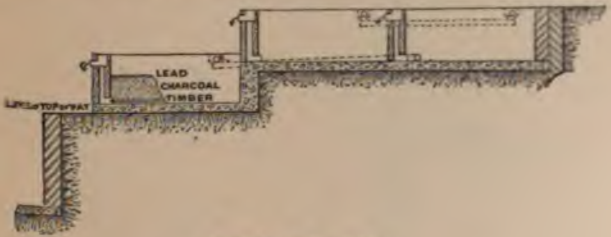
* *An Epitome of Chemistry*, by Wm. Henry, London, 1801, p. 95.



Scale, $\frac{1}{16}$ inch = 1 foot.
Fig. 65.

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* *Proc. Inst. Civil Eng.*, vol. cxlii. (1900), p. 302. Fig. 65 is reproduced by kind permission of the Institution of Civil Engineers.

† *Eng. and Mng. Journ.*, Dec. 17, 1898, p. 724.

‡ *Australian Mining and Metallurgy*, p. 292.

The exact action of the charcoal has not been fully demonstrated. It acts slowly on cold solutions, and its action is not rapid even at boiling point. It is under the disadvantage that it does not destroy free chlorine, which must therefore be expelled by boiling or by passing a current of air through the liquid before the precipitation of the gold is begun. Mr. Davis states that 240 parts of charcoal are required for the precipitation of 19¼ parts of gold. The prevailing opinion is that the hydrogen and hydrocarbons remaining in the charcoal are the active agents in the precipitation, hydrochloric acid and free gold being formed.

Copper Sulphide and *Iron Sulphide*, either precipitated or fused, have been proposed,* but apparently not used for any length of time in practice. The sulphides are not mixed with the solution of gold chloride, but contained in filter-boxes, through which the solution is passed. The free chlorine is destroyed by the sulphides, and the iron or copper gradually removed. *Iron* or *Copper Shavings* were also proposed by the author† as a precipitant for gold, but have not been used.

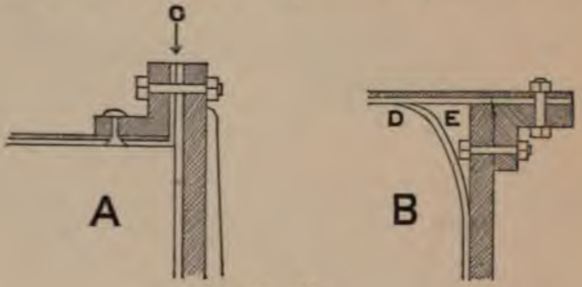

Fig. 66.

Electrical Precipitation.—It is stated‡ that this was tried by W. Greenawalt, who used over again in the barrel the solution from which the gold had been precipitated. In this case the imperfection of the precipitation was of little moment, but in general solutions become too highly charged with salts of lime and base metals to be conveniently used more than once.

Modern Practice in Barrel Chlorination.§—In 1890, improved methods were introduced by J. E. Rothwell at the Golden Reward Mill, S. Dakota, for treating ores not concentrates, and mills were erected later at Cripple Creek and other places in Colorado and Dakota. Further slight improvements were made from time to time, and in 1896 Rothwell stated* that 1,000 tons of ore a day were being treated by barrel chlorination in America, but the cyanide process is now generally preferred, and the industry seems to be languishing.

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* *Eighth Report Cal. State Min.*, 1888, p. 839; *Eng. and Mng. Journ.*, Jan. 24, 1891; *Mining Journ.*, March 11, 1893.

† *Mining Journ.*, March 11, 1893.

‡ *Eng. and Mng. Journ.*, Aug. 15, 1903, p. 244.

§ *Mineral Industry*, vol. i. (1892), p. 233; vol. v. (1896), p. 261; vol. x. (1900), p. 359.

The ores are crushed dry in rock-breakers, followed by two or three sets of rolls, the finished product passing through 12- to 20-mesh screens. The ore is dried before being finely crushed. The roasted and cooled ore is then charged into large barrels, which had a capacity of 3½ tons at the Golden Reward, but hold as much as 12½ or even 18 tons in later practice. The 12½-ton barrels are 16 feet long inside and 5 feet 6 inches in diameter. The trunnions are 16 inches in diameter. The barrels consist of a steel shell of ⅝-inch thickness, lined with ½-inch pure lead of 24 lbs. to the square foot. The lining is bolted to the shell. There are two oval charging-holes, each 11 × 16 inches. The cast-iron heads are 1¾ inches thick.

An enlargement of part of the end of a good type of chlorination barrel is shown in A, Fig. 66; in B, the construction of Rothwell's barrel is shown. The shaded parts of A and B are made of iron (the barrel-head and the flange being cast iron, and the cylinder, boiler-steel); the lead lining is not shaded. In A the lead lining is burnt-on to the flanged cylinder and to the head respectively, and the head then firmly bolted to the flange. The lead-joint, C, is made tight by the blows of a hammer on a blunt chisel directed on the surface of the lead, in the direction shown by the arrow. When repairs are needed inside the barrel, the cast-iron head is removed and the work thus easily done. In the other barrel the lead lining of the end is burnt-on to the lining of the cylinder at D, leaving a hollow space, E. It is stated by Dennes that, in consequence of the existence of this space, the lead-joint is often broken when the pressure in the barrel is great, and leakage then occurs; moreover, repairs must be effected by a workman crouching inside the barrel.

The chlorination barrel is also the washing and leaching vessel; this is arranged by placing a supporting diaphragm, for a filtering medium, to form the chord of an arc of the circle of the barrel. The filter or diaphragm, as it is called, was at first made of asbestos cloth, resting on a framework consisting of oaken planks, each 11 inches wide and as long as the barrel, and 2 inches thick. The area of the filter in the 12½-ton barrel is 4 by 16 feet, or 64 square feet. The planks were grooved, and the filter-cloth rested on the sharp ridges between the grooves, the surface being almost entirely available for filtration. Above the asbestos cloth was placed an open wooden grating, and the whole was held in place by cross-pieces, the ends of which rested under straps bolted to the inside of the shell. The cloth lasted for from 15 to 18 charges, or from 2½ to 3 days. In later practice the asbestos cloth was replaced by 4 lbs. sheet-lead, with fine perforations, supported by sheet-lead ⅜-inch thick with ⅜-inch perforations, and enclosed between wooden gratings. The thin sheet-lead lasts from 6 to 60 charges. A sand filter has also been used inside the barrel.

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* *Mineral Industry*, vol. v., p. 261.

The barrel is charged by first filling the space under the filter with water, which at the same time is allowed to pass through the filtering medium, and wash it; then the required quantity of water is put in above the filter. The sulphuric acid is then poured into the water, through which it sinks in a mass to the bottom, without mixing with it; the ore is then charged in, as follows:—The hoppers are furnished with 2 shoots, one for each charging-hole. The ore is let fall through these shoots alternately, the hole through which ore is not being passed serving as an air-vent. Meanwhile the bleaching powder has been weighed-out and placed in two small kegs. When the ore has all been introduced into the barrel, a workman, stationed at each charging-hole, hollows out a space in the surface of the dry ore with his hands, and, emptying one of the kegs into the barrel, closes up the charging-hole as quickly as possible. If all these operations have been conducted rapidly without a hitch, there is no immediate evolution of chlorine, but, if some time is suffered to elapse after charging-in the ore, the acid liquid, thoroughly stirred-up and mixed by the fall of the ore into it, gradually rises through and wets the charge, and the bleaching powder, falling on ore which has been wetted with acid, gives off copious fumes of chlorine before the cover-plate can be screwed on.

After the chlorination is complete the barrel is stopped, so that the filter assumes a horizontal position; the hose is attached to one of the outlet pipes, and after waiting for a few minutes to allow the pulp to settle, as recommended by Rothwell, the valve is opened and the solution allowed to run out, the pressure of gas left in the barrel being enough to start the leaching. A hose is also attached to the inlet pipe, and water is pumped in under a pressure seldom exceeding 40 lbs. per square inch, the air in the top part of the barrel being compressed and forming an elastic cushion. By washing in this manner, no chlorine is allowed to escape into the building, as it is all absorbed by the water. If necessary the leaching is suspended at intervals, and the barrel is again revolved for a few minutes, so that its contents are thoroughly mixed-up together again. In this way the formation of permanent channels in the ore is prevented, and perfect leaching is ensured. The wash-water coming from the barrel is tested for gold with sulphuretted hydrogen. The full charge of ore is said by Rothwell to lie to a depth of 38 inches on the filter in his 12½-ton barrel, and the average time of leaching to be 2½ hours. The amount of water required for leaching is about 120 gallons per ton, besides the 100 gallons per ton contained in the charge.

The tailings are discharged into a car which will hold the whole charge of ore and water, and then run out of the building; or, if water is abundant, they are discharged into a sluice, and washed away. The filter-cloth is washed clean by a jet of water under pressure directed successively to all parts of it. This water is discharged by revolving the barrel.

The solution coming from the barrel, passes first to a tank with a gravel filter and then to the settling tanks, which are of sufficient capacity to allow several hours for settling. The clear liquid from these is forced up by air pressure into the precipitating vats.

At the Golden Reward Mill there were two precipitating tanks for each barrel. They were 6½ feet in diameter and 10½ feet deep, fitted with covers, and were constructed of wood and lined with lead of 8 lbs. to the square foot. The precipitated sulphides were collected in a filter press, situated on the ground-floor, over 25 feet below the precipitating vats. The press had twelve chambers of 19 inches, and had a filtering area of 57 square feet. Two lead pipes connected the precipitating tanks with the press, one leading from the bottom of the tank and the other from a point in the side about 4 inches above the bottom. The precipitating gas generators were cast-iron cylinders of the same size as the slime filters; only the sulphuretted hydrogen generator was lined with lead.

It is stated that solutions containing gold, copper, arsenic, antimony, &c., yield a precipitate containing little but sulphide of gold, if precipitation is stopped at the right moment, and if free acid is present. The sulphide cakes in the presses are compressed by blowing air through them, and are turned out in the form of lumps, which break up into powder when they are touched; they are then dried, roasted in muffle furnaces in iron trays, 44 inches long, 24 inches wide, and 4 inches deep. The roasted precipitate, which may contain as much as 75 to 80 per cent. of gold, is fused with borax, nitre, carbonate of soda and sand.

At the Golden Reward there were four barrels, each treating 6 charges in 24 hours, or a total of over 90 tons per day. The cost of treatment is given below. It is stated that 92 per cent. of the gold was being extracted in 1894.*

The cost of barrel chlorination in 1891, at the Golden Reward Mill, is given by John E. Rothwell, as follows:—†

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* *Eng. and Mng. Journ.*, Jan. 27, 1894.

† *Ibid.*, Mar. 25, 1893, p. 269.

	July.	August.	September.
Amount of ore treated, .	1,430 tons.	1,195 tons.	1,513 tons.
,, treated per day, .	46·1 ,,	38·5 ,,	50·4 ,,
Cost per ton—			
Milling, . .	$1.49	$1.40	$1.44
Roasting, .	1.53	1.35	1.37
Chlorination, .	1.76	1.67	1.65
Office, salaries, .	0.40	0.47	0.37
Construction and repairs, .	0.36	0.93	0.21
	$5.54	$5.82	$5.04

The total cost given above includes all the working expenses, but is exclusive of interest on capital, taxes, insurance and amortisation (*i.e.*, extinction of capital by a sinking fund, which covers the depreciation of plant, &c.).

The roasting was at that time done in Brückner cylinders, and its cost was afterwards reduced to about 70 cents per ton by the adoption of a modified White-Howell furnace, and in January, 1894, the total cost of treatment per ton was stated to be only $3.77. In 1896 the cost of roasting in Brown's Horse-shoe furnaces was stated to be less than 50 cents per ton.

The amount of chemicals used in the chlorination and precipitation departments, in 1891 is given below :*—

	JULY, 1891.		AUGUST, 1891.		SEPTEMBER, 1891.	
	Lbs. per ton of Ore.	Cost in Cents.	Lbs. per ton of Ore.	Cost in Cents.	Lbs. per ton of Ore.	Cost in Cents.
Sulphuric acid, .	26·82	45·6	25·14	44·0	24·77	43·3
Chloride of lime, .	11·04	35·8	10·17	33·1	10·09	32·8
Crude sulphur, .	0·37	1·14	0·31	0·9	0·29	1·2
Iron sulphide, .	0·77	4·54	0·79	4·0	0·84	4·2
Total cost, .	...	87·1	...	82·05	...	81·5

The working costs at Cripple Creek in 1900 were given by J. E. Rothwell as $3.53 per ton.†

Bromination at Rapid City, Dakota.—The following details were supplied to the author by D. Dennes. The ore which was treated in 1892 contained 6 or 7 per cent. of sulphur, a similar amount of arsenic, and some iron, but little or no copper. It was roasted in White-Howell and Brückner furnaces, and charged into lead-lined revolving barrels, 8 feet long and 4½ feet in diameter, and having a capacity of 3½ tons. Hot water was introduced into the barrel before charging-in the ore, only 66 gallons per ton or 33 per cent. of the weight of the ore being used.

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* *Loc. cit.*

† *Mineral Industry for 1900*, p. 370.

This seems too small a quantity for efficient treatment. The barrels had cast-iron trunnions 12 inches in diameter, and were revolved at the rate of twelve turns per minute by a worm and wheel. They were made of boiler iron, ⅜ inch thick, and were lined with thick lead of 24 lbs. to the square foot on the ends, and 18 lbs. to the foot on the cylinder.

After adding the ore and water, the required amount of bromine was poured in, and the barrel closed at once. Little inconvenience is stated to have been caused to the workmen by the bromine fumes which were evolved owing to this crude method of manipulation; the men's eyes only were slightly affected. From 6 to 13 lbs. of bromine were added to the charge of 4 tons, the amount being adjusted so that a very slight excess of free bromine remained at the end of the operation. Chlorine generated from sulphuric acid, and bleaching powder was used for several months before the introduction of bromine, but the cost was greater and the tailings only ¼ dwt. poorer in the former case. Moreover, the filter cloths lasted longer, the health of the workmen suffered less injury, and the gold precipitate was purer when bromine was used. The barrel was revolved for from thirty minutes to one and a-half hours, and then discharged into the leaching vat below. This was 7½ feet in diameter and 3 feet deep, made of cast iron lined with lead, and had a strongly ribbed cover, so that it was capable of withstanding an internal pressure of 100 lbs. to the square inch. Internally, it tapered slightly upwards. The sides of the vat did not rest on the bottom, but were supported by columns direct from the floor. The bottom was not connected with the rest, but was supported by a hydraulic ram. On this true bottom there was a filter-bed and false bottom. The filter-cloth consisted of gunny-sacking. The vat was worked as follows:—When a charge was about to be introduced, the bottom was raised by the ram and pressed tightly against the sides of the vat, a good joint being made by a thick rubber ring. The hydraulic pressure was more than enough to overcome the air pressure subsequently applied. The charge was then introduced, the cover replaced and fastened down tightly by screw-bolts, and air was pumped into the vat above the surface of the charge. Water was also introduced by means of a lead pipe extended round the vat at about the surface of the charge. This pipe was pierced with a number of small holes, by means of which jets of water were thrown upon all parts of the ore surface. The air pressure, which never exceeded 60 lbs. to the square inch, forced the wash-water through the charge very quickly. The solution was of a strong ruby-red colour, due to the presence of free bromine, and of the bromide of gold, the colour of which in solution is very intense. There was no need to test the issuing liquid, as its colour was found to be a perfect guide to an experienced eye. When it had become colourless, the water supply was shut off and the charge dried by pumping air through it. This was necessary, as the company was not allowed to sluice the tailings into the river. When no more water could be driven out, the air-pressure was let off, and the bottom of the vat lowered until the top of the filter-bed was just clear of the sides of the vat. The bottom was then drawn sideways from below the vat by means of a second hydraulic ram, and the charge fell into a large ore-car below, having a capacity of 4 tons. This ore-car then ran by gravitation to the dump, and was emptied and drawn back by a wire rope winding on a drum actuated by steam power. It is stated that the leaching occupied only twenty minutes, and the vat was used to leach the charge from the two barrels, which discharged into it alternately. The vat was so rapid that it was always waiting for the barrels, and 100 tons per day could be leached by it.

The liquid was forced into large, lead-lined wooden precipitation vats, each 20 feet × 10 feet × 6½ feet, which were placed outside the mill, buried under 2 feet of manure to keep them warm in winter. The precipitation was effected by the successive use of sulphur dioxide and sulphuretted hydrogen, applied in exactly the same manner as at the Golden Reward Mill. The collection of the sulphides was also effected in a similar manner. When dried, the sulphides were put, together with the necessary amount of borax, into a small barrel, revolving by machinery, and were mixed thoroughly. The mixture was then transferred by a scoop to a red-hot clay crucible (size No. 100) in the furnace, additions being made at intervals until the crucible was full of bullion and molten slag. The latter was very rich and full of shots of metal. It was stored and eventually sold to the smelters, but owing to the difficulty in sampling it, it would have been better to fuse it with lead and to sell the latter.

The air-compressor used was 12 inches in diameter, and had an 18-inch stroke; its maximum velocity was eighty revolutions per minute. A large air receiver, 16 feet long and 4 feet in diameter, was used between the pump and the leaching vat. The maximum air pressure in the receiver was 80 lbs. per square inch. From 60 to 75 per cent. of the gold in the ore was extracted, the tailings usually containing over 5 dwts. of gold. The reason for this unsatisfactory result is said by Dr. L. D. Godshall (*Eng. and Mng. Journ.*, Jan. 6, 1894) to lie in the coarseness of the crushing, a 10-mesh sieve being used. When a 30-mesh sieve was used, only about 2⅓ dwts. were left in the tailings, and the product was easily leached. It seems possible that inefficient leaching, due to the formation of channels in the ore, may have been another cause of loss, but the chief cause of the failure was doubtless the fact that the amount of bromine used was inadequate, the endeavour to reduce the costs of the process being fatal to success.

CHAPTER XV.

CHLORINATION: THE VAT-SOLUTION PROCESS.

The Munktell Chlorination Process.—This method was devised by W. Munktell, who seems to have worked it out without having visited either the vat or the barrel chlorination works already established in other parts of the world. The process enjoys the distinction, according to the published accounts, of having been worked at a profit. It was first used at the Fahlun Copper Works, Sweden, and was afterwards introduced into Hungary, where it was worked at Brade and at Boitzas. The process as worked at Fahlun may be briefly described as follows:—The ore is roasted at a low temperature with the view of obtaining the copper in the form of sulphates. If silver is present, salt is added in the roasting furnace. The calcined ore is then placed in false-bottomed, wooden vats and leached with hot water, followed, if necessary, by hot, dilute sulphuric or hydrochloric acids, by which all the copper and much of the iron are removed from the ore. The solutions thus obtained are run through tanks containing scrap iron, by which the copper and any silver in solution are precipitated. The solution of ferrous sulphate may be saved and used subsequently to precipitate the gold. The residues in the tubs are now in a condition to yield up their gold readily, and are accordingly treated by a solution of 0·6 to 0·7 per cent. of chloride of lime (bleaching powder) in water, mixed with an equal volume of dilute hydrochloric acid of specific gravity 1·002 or 1·003, or of dilute sulphuric acid. These solutions are mixed in troughs just before they flow into the ore-vat. Chlorine is slowly generated by the action of the acid on the hypochlorite of lime, and, being stated to be partly present in the nascent state, attacks the gold vigorously in spite of the extreme dilution of the solution, while there is very little smell of the gas above the vats. It would appear, however, that the generation of chlorine would soon be completed, and the dissolving action would then be carried on by an aqueous solution containing about 0·1 per cent. by weight of chlorine, or about 0·3 volume of chlorine gas in one volume of water. The liquid is passed through the ore until gold ceases to be dissolved, after which the tailings are thrown away. The solution is heated to 160° F. by steam, and precipitated by ferrous sulphate. The collection of the gold is expedited and ensured by adding acetate of lead to the solution; by this device lead sulphate is precipitated with the gold, and, in settling to the bottom, carries the particles of precious metal with it. This process was in continuous operation at the Fahlun Copper Works from 1885 to 1888. In that period, 1,500 tons of gold ore and the tailings from 29,000 tons of copper ore were subjected to treatment. It is stated that in the year 1886, the tailings from 14,000 tons of copper ore were treated with the following results:—

AVERAGE AMOUNT OF GOLD IN TAILINGS.

Before treatment,	41·82 grains per ton.
After ,,	4·04 ,,

COST PER TON.

Chloride of lime (6 lbs.),	5d.
Sulphuric acid (8·37 lbs.),	1d.
Plumbic acetate and other reagents,	¾d.
Fuel for steam,	1¼d.
Labour,	1d.
Total,	9d.

The low cost assigned to labour is very remarkable. These tailings had, of course, been previously crushed and roasted before being treated as above.

In the same year 960 tons of gold ore were treated at a cost of 12s. 2¾d. per ton, the ore containing 523·62 grains of gold per ton before treatment, and 6·02 grains per ton after treatment, so that no less than 98·85 per cent. of the gold was extracted.

The following details concerning the practice at Brade, Hungary, may be added.* In addition to gold, the ores contain iron pyrites, barytes, zinc-blende, antimonial minerals and argentiferous galena, and also in some cases calcite and carbonate of magnesia. The concentrates, which are subjected to treatment by the Munktell process, contain 0·86 oz. of gold and 4 ozs. of silver per ton, and 36 to 40 per cent. of sulphur.

The oxidising roasting takes twenty-eight hours, after which 5 per cent. of salt is thrown upon the ore and mixed thoroughly with it, and four hours later the charge is drawn from the furnace and allowed to cool slowly in a covered-in brick pit. The loss by volatilisation is, however, considerable, and it is proposed to build seven-storey furnaces. The leaching vats are made of wood, lined with lead, and are 3 metres broad, 5 metres long, and 0·75 metre deep, and hold 10 tons, the charge being 0·5 metre deep. There is a false bottom and filter-bed of quartz as usual. The ore is leached with five different solutions in succession, viz. :—(1) Warm water, 2,500 gallons being required for the charge: Chlorides of copper and zinc and about 25 per cent. of the silver are removed in solution. (2) A solution containing 2 per cent. of hyposulphite of soda, 1,600 gallons being used; this dissolves the rest of the silver. (3) Dilute sulphuric acid, 2,700 gallons being used to remove the oxides of iron. (4) Weak solutions of bleaching powder and sulphuric acid to dissolve the gold; and (5) Cold water to wash the ore. The solutions are all preserved separately. The leaching takes in all nine or ten days, the chlorination alone occupying three days. The gold and silver are precipitated from their solution by sodium sulphide, and the precipitate is dried, pressed, roasted, and melted down.

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* *Proc. Inst. Civil Eng.*, Session 1891-92.

The expenditure per ton of concentrates is as follows:—

	s.	*d.*
Salt (110 lbs.),	1	7
Sulphuric acid (102 lbs.),	6	0
Bleaching powder (25 lbs.),	3	7
Hyposulphite of soda (14·7 lbs.),	1	10
Fuel,	8	0
Labour,	3	8
Amortisation,	1	7
Total,	26	3

The percentage of extraction is not given.

Somewhat similar treatment is followed at Bovisa, in N. Italy, where about 25 tons per day of pyritic ore, containing 34 per cent. of sulphur, 10 to 12 per cent. of arsenic, and 0·6 to 0·7 oz. of gold per ton, are roasted dead (the arsenic being recovered) and chlorinated in lead-lined wooden vats holding 10 tons each. Weak solutions of bleaching powder and sulphuric acid are allowed to pass slowly through the ore, and the total time of treatment is three days. From 85 to 87 per cent. of the gold is recovered.*

The Cassel-Hinman Bromine Process.—The ore at the Nellie Bly Mill, Magnolia, Colorado, was treated in open vats by nascent bromine generated in the interstices of the ore. A mixture of bromide and bromate of soda or lime is prepared by adding bromine to a nearly saturated solution of soda or milk of lime. A solution in water of these salts is said to yield the whole of its bromine in the free state, when treated with an acid. The reactions are expressed thus:—

$$(1)\quad 6NaOH + 3Br_2 = 3NaBr + 3NaBrO + 3H_2O$$

On heating or standing, this becomes

$$(2)\quad 6NaOH + 6Br = 5NaBr + NaBrO_3 + 3H_2O$$

$$(3)\quad 5NaBr + NaBrO_3 + 3H_2SO_4 = 3Na_2SO_4 + 3Br_2 + 3H_2O$$

Under similar circumstances only part of the chlorine would be liberated.

The ore which contained from 10 to 15 dwts. of gold was crushed by rolls and roasted in a Brown Horseshoe Furnace, after which it was fed into open lead-lined vats, provided with the usual filter-bed at bottom. The solution was then run over the ores, and the filtrate containing the gold, bromine, and soluble bromides was run into a closed tank, where chlorine was added to liberate whatever bromine had combined with the bases of the ore. The loss of bromine was stated to be only about ½ lb. per ton of ore treated.

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* F. Clerici in *Eng. and Mng. Journ.*, Aug. 29, 1896.

The gold solution, after the bromine had been separated from it by an air current, flowed into a precipitating tank, and was precipitated by sulphuretted hydrogen.

One of the advantages gained by using bromine instead of chlorine is that solutions are obtained much more free from base metals which interfere with precipitation and make the bullion base. However, the process was soon abandoned.

The Permanganate Chlorination Process.—This process differs from the ordinary treatment of ore in vats by chlorine water only in the use of a different solution. M. Etard originally proposed a solution containing 45 lbs. of strong commercial muriatic acid, and 12 ozs. of crystals of permanganate of potash. Professor Black, of Otago University, as an alternative suggested 12 lbs. of common salt, 14 lbs. of sulphuric acid, and 6 or 7 ozs. of permanganate crystals per 100 gallons of water. These solutions smell slightly of chlorine, but it is claimed that in the absence of gold or other material attacked by chlorine, practically no free chlorine is produced. Both solutions dissolve gold at less than one-tenth the rate of a saturated solution of chlorine in water (see above, p. 12).

The method was tested by the treatment of 2,000 tons of ore at the Bethanga Mill, Victoria, in 1900.* The ore consisted mainly of pyrites, mispickel, and chalcopyrite with some blende containing an average of 25 per cent. of sulphur and 9 per cent. of arsenic. It was crushed dry by a ball mill to 20 mesh and roasted in hand-worked reverberatory furnaces, 40 lbs. of salt per ton being added a few minutes before withdrawal. Wooden vats, 5 feet deep, holding 17 tons of roasted ore were used for chlorination. The solution contained 110 lbs. of commercial sulphuric acid, 90 lbs. of common salt, and 12 lbs. of potassium permanganate (per 1,000 gallons?) The average time of treatment was 140 hours, the tailings being tested by panning, and finally leached with hot water. The ruby colour of the solution was used to gauge its solvent strength. The average contents of the roasted ore during one month were gold 2·313 ozs., silver 1·63 ozs. per ton, and the tailings contained an average of 0·147 oz. gold per ton, or nearly 3 dwts. This corresponds to an extraction of 93·7 per cent. of gold. About 500 tons of roasted ore were treated per month at a cost for chlorination, according to E. Harris, of 5s. 11·6d. per ton (?of the original ore). The cost of roasting was 12s. 7d. per ton. According to D. D. Rosewarne,* the cost of chlorination, using permanganate, during four months averaged 11s. 5d. per ton, and using chlorine at the same time the average cost was only 7s. 4d. per ton. On this exhaustive test the permanganate process was abandoned.

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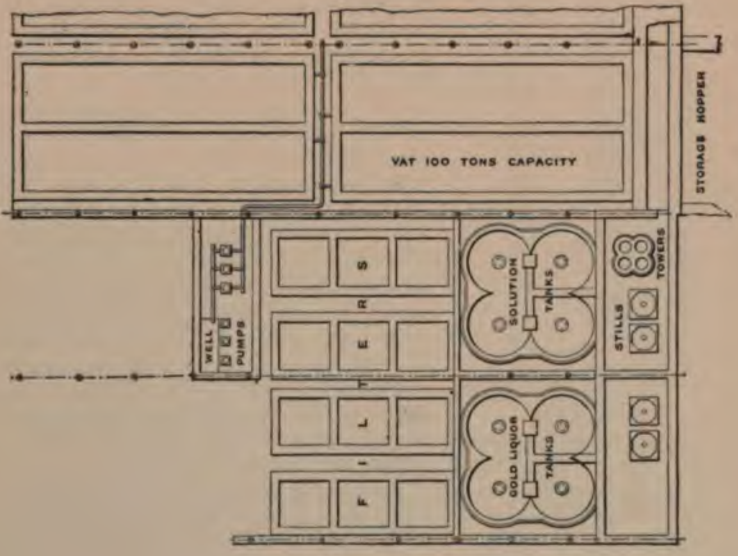
* "The Permanganate Chlorination Process at Bethanga," by E. Harris. *Inst. of Mng. and Met.*, Dec. 15, 1904.

Chlorination at the Mount Morgan Mine, Queensland.† —This is the largest and most important chlorination mill in the world. The method of treatment is the original Munktell process, adapted and improved. It consists in subjecting the ore in open vats to the action of a solution of chlorine in water. It was adopted, after chlorination in 1-ton barrels had been used for about a year, in 1887-1888.



Scale, 1 inch = 40 feet.
Fig. 67.‡

The ore from the upper levels, now chiefly in the form of mine tips, consists of sinter, kaolin, quartz, and ironstone, with a small percentage of sulphur. The ore from the lower levels consists of hard quartz, heavily charged with pyrites. The gold in the latter is believed to be mainly in the form of tellurides, and after roasting all the ores contain only finely-divided gold. The ores are treated in different plants by different methods.

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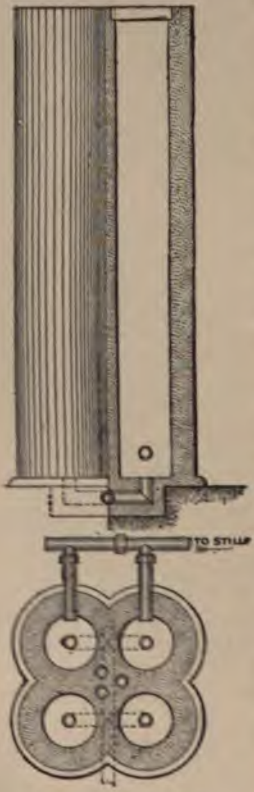
* *Institution of Mining and Metallurgy*, report of meeting on 15th Dec., 1904, p. 25.

† See E. W. Nardin, *Proc. Inst. Civil Eng.*, vol. cxlii. (1900), pp. 297-307; and *Australian Mining and Metallurgy*, by Donald Clark, pp. 265-294. Figs. 67-70 are reproduced by kind permission of the Institution of Civil Engineers.

‡ *Proc. Inst. Civil Eng.*, vol. cxlii.

At the "West Works" plant, a plan of which is shown in Fig. 67, over 100,000 tons of low-grade ore from the upper levels are treated annually. The ore is crushed dry in Krupp ball-mills (see above, p. 197), to pass a 20-mesh screen, roasted in revolving cylindrical furnaces and stored in main hoppers higher than the vats. The roasting is mainly to dehydrate the ore, so as to improve the speed of leaching, which is otherwise excessively slow. A small percentage of sulphur is also removed. The vats are 60 feet long, 12 feet 6 inches wide, and 5 feet deep, and hold 100 tons of ore. They are made of concrete lined with neat cement, which in turn is covered with pitch and tar applied hot. There are sixteen of these vats. The filter bed is 20 inches thick, consisting of 16 inches of sand and gravel resting on perforated planks supported by wooden bearers. The vat has a fall of 3 inches from one end to the other for draining off the solution. The depth of the bed of ore is little more than 3 feet. The ore is tipped in from trucks and roughly levelled.



Scale, ½ inch = 1 foot.
Fig. 68*.

The chlorine is generated in ordinary flagstone stills from sulphuric acid, manganese dioxide, and common salt. The acid is made on the spot from imported sulphur. The manganese and salt are crushed and thoroughly mixed before charging into the still. The chlorine gas is passed into scrubbing towers, see Fig. 68, each 2 feet 3 inches in diameter and 20 feet high, filled with glass bottles and old assay crucibles, always kept wet by trickling water. The solution of chlorine contains about 600 grains of chlorine to the cubic foot, or 0·14 per cent. of chlorine. The weight of chlorine used is about 2½ lbs. per ton of ore, which corresponds to about 30 cubic feet of solution per ton. The solution is stored in solution tanks (see Fig. 69), closed chambers with concrete domes, whence it flows through a 6-inch earthenware main to the vats. There are four stills, four towers, and four solution tanks, the latter having a capacity of about 950 cubic feet each.

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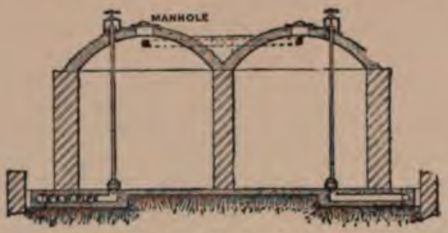
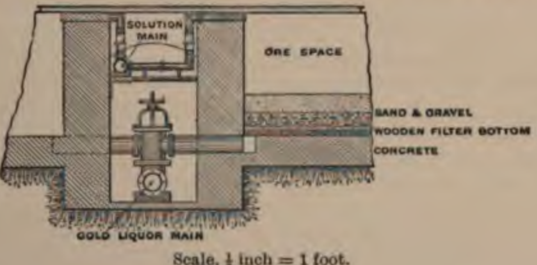
* *Op. cit.*

The solution is allowed to run on the charge, from 6-inch earthenware mains (see Fig. 70), until the vat is full, when the valve below the false bottom is opened and the vacuum in the pump main of about 5 lbs. per square inch draws the liquor through. Fresh solution is run on to keep the ore covered, and this is continued until the gold is dissolved, which appears to take about thirty-six hours. During the first few hours all the chlorine is absorbed by the ore which becomes hot, doubtless from the attack on ferrous and basic salts as well as sulphides left in the ore after roasting. After about fifteen hours, the liquor issuing from the vat is found to contain gold, which gradually increases in amount.



Scale, $\frac{1}{16}$ inch =1 foot.
Fig. 69.[*]



Scale, $\frac{1}{4}$ inch = 1 foot.
Fig. 70.[†]

When the liquor gives a good black precipitate with ferrous sulphate, and smells strongly of chlorine gas, it is assumed that all the gold is dissolved, and the vat is then said to be "chlorinated," and wash-water is turned on and passed through the ore until no gold can be detected in it by means of ferrous sulphate. The ore is then discharged by means of shovelling into trucks, though it is intended to replace this by a steam dredge. The solution of gold is passed through charcoal filters as described on p. 268.

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[*] *Op. cit.*

[†] *Op. cit.*

At the "Top Works," the ore treated is hard compact quartz heavily charged with iron pyrites. It contains an average of 4 ozs. gold per ton and 11 per cent. sulphur, which after roasting is reduced to 0·15 per cent. The ore is roasted in a Richards' shaft furnace (see p. 230) at a cost of about 5s. per ton. The amount of solution required is larger than that at the West Works, but the time of treatment is about the same, as the ore is more porous and the washing rapid.

The following are the chief items in connection with the chlorination at Mount Morgan during May, 1898:—*

	West Works.	Top Works.
Tons chlorinated, . . .	9,852	1,534
Average assay of ore, . . .	11 dwts. 13 grs.	4 oz. 4 dwts. 10 grs.
,, ,, residue, . .	22 grs.	4 dwts. 6 grs.
,, extraction, . . .	92·06 per cent.	95 per cent.
,, time under "solution,"	83 hours.	64 hours.
,, ,, of washing, . .	35 ,,	6 ,,
,, weight of chlorine per ton of ore, . .	2·51 lbs.	11·47 lbs.
Cost per lb. of chlorine, . .	5·85d.	5·53d.
Weight of chlorine per cubic foot of "solution," . . .	620 grains.	599 grains.
Total cost per ton of ore for chlorinating and precipitating, .	4s. 4·58d.	...

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* *Op. cit.*, pp. 304, 305.

CHAPTER XVI.

THE CYANIDE PROCESS.

Introduction.—The cyanide process will always be indissolubly connected with the names of its discoverers, J. S. MacArthur and the Forrests. The solvent action of potassium cyanide on metallic gold and silver has long been known, as is pointed out in Chapter xvii., but it was believed that the use of an electric current in conjunction was needed to quicken the action, which was otherwise too slow to be of any practical value. Until the surprising results of the MacArthur-Forrest process, as it was called for some years, were made public, metallurgists were all disposed to condemn the use of cyanide as a solvent for gold in ores as being chimerical, on account of the instability of potassium cyanide, combined with its high cost and poisonous character.

The first attempt at the direct extraction of gold by the use of cyanide was made by J. H. Rae, who took out a patent in the United States in 1867 for a process dependent on the removal of gold and silver from their ores by the combined action of a "current of electricity and of suitable solvents or chemicals," such, for instance, as cyanide of potassium, the gold being simultaneously precipitated on copper plates by the electric current.

In 1885, J. W. Simpson, of New Jersey, proposed to treat ores containing gold, silver, and copper by a solution containing 3·0 per cent. of potassium cyanide and 0·19 per cent. of ammonium carbonate. Copper was to be dissolved at the same time as the gold; if silver was present also, an addition of common salt was made. The inventor appears to have believed that, by using carbonate of ammonia, the necessity was obviated of employing an electric current, in conjunction with cyanide of potassium, in order to dissolve the gold. The precipitation of gold was effected by "a piece or plate of zinc." The process was not tried on a large scale.

In 1886, however, a series of experiments on wet processes of treating gold ores was begun by J. S. MacArthur and R. W. and Wm. Forrest in Glasgow, and it was entirely owing to their energy and skill that cyanide of potassium was successfully applied in practice to the treatment of gold ores. Their process [illegible] in attacking gold and silver ores by dilute [illegible] ss than 1 per cent. of KCy, caustic soda [illegible] ores rendered acid by the oxidation of pyrites, and then in precipitating the precious metals by zinc shavings. This process has now passed into use in all parts of the world. Its success is complete on many ores, and its extension has become very great, partly at the expense of the chlorination process, but mainly in treating the tailings of ores which have been crushed and amalgamated. These tailings were formerly run to waste. In a still larger number of cases, ores are treated by amalgamation and cyaniding which were formerly too poor to treat at a profit by any known process. The chief advantage of the cyanide process over the chlorination process is that roasting is not necessary, even if sulphides are present; this is a most important point in the treatment of low-grade ores, especially in places where fuel and labour are costly. Moreover, silver as well as gold is extracted by cyanide solutions. If tellurides are present, ordinary cyanide is inefficient, and the ores must either be roasted or treated with bromo-cyanogen. Messrs. MacArthur and Forrest, in the course of their investigation on wet methods, became dissatisfied with chlorine as a solvent, owing to its energetic and preferential action on sulphides of base metals and other bodies, and its inapplicability to ores containing silver. They desired to find a solvent which would exercise a selective action in favour of the precious metals, instead of other substances. With this object in view, they experimented with a number of solvents (such as ferric bromide, ferric chloride, &c.), and finally decided that potassium cyanide possessed advantages over all other substances. They showed that it was essential to use very dilute solutions, which do not energetically attack pyrites and other base minerals, but readily dissolve gold and silver.

The process comprises four distinct operations, viz.:—(1) Preparation of the ore for treatment. (2) Solution of the gold in potassium cyanide. (3) Precipitation of the dissolved gold by means of zinc; and (4) Conversion of the precipitated gold into bullion by fusion.

1. Preparation of the Ore.—The ore is crushed to a suitable degree of fineness, which depends partly on the condition of the gold and partly on the nature of the gangue. In the Transvaal, where tailings are treated, the ore is crushed in stamp batteries previous to amalgamation, the screens being usually about 20 or 25 mesh. Here, and in all similar cases, part of the ore is reduced to "slimes" or very finely-divided material which interferes with percolation. The tailings were formerly allowed to settle in pits or dams, when part of the slimes passed off suspended in the water. The sands, however, often contained too much slimes for efficient leaching, and the practice has been superseded generally. The tailings are now usually de-watered and partially classified in spitzkasten, and the overflow of slimes either allowed to accumulate in settling pits or else treated separately with cyanide. The material discharged from the bottom of the spitzkasten which still contains some slimes is sometimes passed into deep "intermediate vats" by means of Butters and Mein's distributor, an automatic revolving apparatus working on the principle of the automatic garden sprinkler. Pipes of several different lengths conveying the pulp revolve round a central spindle, and spread the pulp evenly over the surface of the water with which the vat is filled. The sand settles to the bottom and accumulates there, while the slimes overflow at the top. Later, however, automatic distributors were discarded at several mines in South Africa and hose-filling resorted to. This gives a more complete separation of sands and slimes, though automatic distribution might be made more satisfactory if the overflow were at the centre.* When the accumulation of sand in the vat reaches nearly to the top, the water is drained off and the ore discharged by shovelling, or by means of machinery, such as the Blaisdell vat excavator, in which steel discs attached to revolving arms push the sand towards the centre of the vat. In either case the sand falls through an aperture in the bottom into ore-cars, or on to a travelling belt below. The sand is now in a suitable condition for leaching, containing only small quantities of slimes. Sometimes the pulp collected in this way is treated directly without removal. In other mills, where the method of "double treatment" of the tailings is employed, the first solution of cyanide is directly applied to these vats, and, after draining, the pulp, wetted with cyanide solution, is transferred to the second treatment vats. As already stated, the usual method is to remove the collected sands to the cyanide vats for treatment.

The slimes overflowing from the intermediate vats are treated with the rest of the slimes.

In places where the separate treatment of slimes is not desirable, either from their small quantity or from other reasons, the slimes and sands are thoroughly mixed before and at the time of charging them into the leaching vat, but the results are seldom quite satisfactory. In India, according to Alfred James,† the siliceous slimes are dried in the sun and then treated in the ordinary way by leaching.

In the direct treatment of ores by cyanide without preliminary amalgamation, it has been usual to resort to dry crushing.

2. **Solution of the Gold.**—This is effected in round vats, constructed of wood, concrete, or iron and steel.

If wood is used it is covered by a coating of paraffin paint, or by a mixture of asphaltum and coal-tar. Concrete and brick vats are not now advocated owing to their great cost and less convenient working, and wooden vats suffer more than iron and

* *Metallurgical Engineering on the Rand*, by J. Yates, p. 98.

† *Cyanide Practice*, p. 82.

steel ones from exposure and from being alternately emptied and filled.* Iron leaching vats are not usually coated inside, as little if any action goes on between the metal and cyanide solutions; occasionally, however, they are coated with tar inside. The steel vats at the Jumpers Deep Mine are 40 feet in diameter and 7 feet deep inside. The sides are of $\frac{1}{4}$-inch steel plate, and the bottoms are $\frac{5}{16}$ inch thick. The plates are rivetted together and strengthened by angle iron at the top and bottom of the vat.

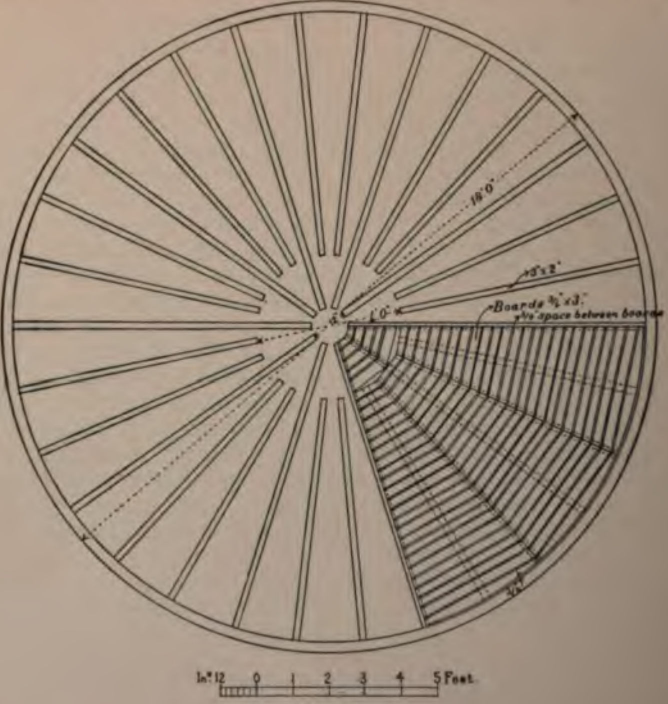

Fig. 71.

Wooden vats are made of staves, 4 to 6 inches wide and 3 inches or more thick, held together by round iron hoops, with bottom planks 3 inches thick, fitting into a slot in the staves. Sometimes in new mining districts square tanks are built, as being cheaper and more easily constructed, but they do not last long and are more difficult to keep tight.†

The dimensions of vats vary with the work to be done. In calculating the capacity of a leaching vat, the volume of a ton of raw ore may be taken at from 22 to 28 cubic feet when dry, and from 20 to 26 cubic feet when wetted down. Tailings sometimes occupy 30 cubic feet per ton or even more. Among very large vats may be mentioned those at the Simmer and Jack Works, in the Transvaal, which are 40 feet in diameter with 14-foot staves, holding 600 tons of ore each. If the material to be treated is such that percolation is difficult, the depth of the vat is kept small, and the diameter made as large as possible. In the direct treatment of dry-crushed ores this is usually necessary, as they always contain some slimes. A depth of 2 or 3 feet of dry-crushed ore is usually as much as can be conveniently leached, and vacuum pumps are often added to expedite the work.

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* *Metallurgical Engineering on the Rand*, by J. Yates, p. 97.

† A. James, *op. cit.*, p. 19.

The vacuum may be obtained by a direct-acting pump, or by the use of a large boiler, in which a vacuum is created by a Westinghouse or other pump. This is, perhaps, the most advantageous course. As soon as the pressure in the boiler falls to about half an atmosphere, it is connected with the aperture of the vat below the filter-bed. The rate of leaching is often doubled by the diminution of the pressure, below the filter-bed, to half an atmosphere, and in some cases it is increased from ½ or 1 inch to 7 or 8 inches of liquid (in the leaching vat) per hour.

The false bottom is usually a wooden framework, constructed of boards pierced with numerous auger holes, or in larger vats of wooden slats crossing each other. The framework is covered by cotton twill, canvas, or cocoa-nut matting, which are not rapidly destroyed by the solution as they are in the chlorination process. Fig. 71 shows a filter bottom for wooden vats used in Colorado. It is from a working drawing kindly supplied by Mr. W. F. Tindall. Thick canvas duck, resting on matting, forms a trustworthy filter-cloth. The filter-cloth is protected by wooden slats which prevent it from being injured by workmen's shovels in clearing out the charge.

An iron pipe communicates with the space below the false bottom, and conveys the liquid to the pumps or to the zinc boxes. The solution does not attack wood or iron; brass and bronzes are attacked and corroded rapidly.

Charging the Ore into the Vats.—The crushed ore is conveyed in cars pushed by hand, running on an overhead railway, and emptied into the vats by tipping. The supports of the railway must not touch the sides or any part of the vats or their supports, as the jarring caused by the running of the cars makes the ore settle down, and so renders the leaching more difficult and tedious.

The vats are filled to within a few inches of the top, and the charge is levelled by means of hoes. The amount of ore charged-in is such that, after the solutions have been applied, the surface of the charge may stand at about 12 inches below the rim. In levelling the ore, the labourer must not step into the vat or forcibly press down the ore, as irregular filtering is produced by this cause. The shrinkage of the charge on the addition of liquid is from 10 to 18 per cent. The ore should be charged-in as dry as possible, but a few per cent. of moisture makes very little difference to the subsequent leaching. As has already been mentioned, the vats are occasionally charged direct with the tailings from the stamp battery, the excess water and slimes being allowed to overflow.

Leaching the Ore.—The charge is now ready for lixiviation. If soluble salts are present in it, a preliminary washing with water is required, while, if decomposed pyrites are present, treatment with an alkaline solution is also necessary, for reasons to be explained when the chemistry of the process is considered. In each case, from 7 to 9 cubic feet of liquid per ton of ore are added from below, rising through the ore and overflowing at the top. The alkali used is either milk of lime or a solution of caustic soda. The former is preferred in South Africa, and latterly has often been charged into the stamp batteries with the ore, the amount being regulated by the requirements of the slimes settlement. After the caustic alkali (if it is used) has been allowed to remain in contact with the ore for a sufficient length of time, it is usually displaced by wash-water, which is added above, while the alkaline solution is allowed to run out at the bottom. A dilute solution of cyanide is then sometimes run in to displace the wash-water and prevent the "strong" solution from being unnecessarily diluted. The amount of weak solution thus used may be from 1 to 3 cubic feet per ton of ore. The strong solution is then added. The liquids are run on to the top of the charge through pipes of 2½ to 4 inches diameter, and are distributed by a large box with a perforated bottom, by which the force of the discharge is broken. The orifice below the filter-bed is closed before beginning to add the liquids, and not opened until the solution stands about 3 inches above the surface of the ore. The strong solution is often allowed to remain in contact with the ore for some hours before percolation is allowed to begin. Sometimes it is necessary to expedite the rate of percolation by means of vacuum pumps, sometimes the lower stopcock is partly closed to prevent the solution from running through too rapidly. The time of treatment with strong solution is often about one or two days.

The strong solution is usually conveyed at once to the zinc boxes, but it was formerly preferred either to raise it and pass it through the charge again (circulation method), or to transfer it to a second or even a third charged vat before precipitating the gold. The advantage of these "circulation" and "transference" methods is that the solutions become much richer in gold than if they were only allowed to percolate through a single charge of ore, and consequently they give a cleaner deposit on the zinc with much less consumption of cyanide, the volume of solution passing through the precipitation boxes being less. At the Mercur Mine the circulation was formerly kept up for from 24 to 240 hours, according to the speed of leaching, the usual time being about sixty hours. At the Robinson Mine 20 tons of solution cover the ore in a 75-ton vat, and were continually pumped back into the same vat for thirty-six hours, and then passed to the zinc boxes.

When the strong solution has drained away, the weak solution is run on. The amount used is often three or four times as much as that of the strong solution, but was formerly less. The oxygenation of the charge, obtained by draining it dry after the completion of the treatment with strong solution, is often necessary to enable the residue of the gold to be dissolved by the weak solution. The weak solution is sometimes drained off, and sometimes displaced by water washes passed through as quickly as possible.

Strength of Solutions.—In the Transvaal, the strong solution usually contains from 0·25 to 0·35 per cent. of cyanide and the weak solution from 0·05 to 0·15. A strength of 0·5 per cent. is, however, necessary in the treatment of some ores, and, on the other hand, it is sometimes better to use no solutions stronger than 0·1 per cent. or even more dilute.

Much stronger solutions than these were used in the early days of the process, but were found to be unnecessary. In 1895, Caldecott* made a series of experiments on a working scale to test the dissolving powers of solutions of different strengths. Five similar vats were charged with 34·03 tons each of fresh slightly-pyritic tailings of uniform composition from the Jumpers battery, containing 100 grains of gold per ton. The treatment that each vat received was exactly the same except that the strength of solution employed was different in every case. To each charge, 15·31 tons of solution were added, and circulated for three days, after which the solution was drained off and displaced by 8·4 tons of wash-water. When the charges had drained dry, they were sampled, the result being as follows:—

	I.	II.	III.	IV.	V.
Strength of cyanide, p.c.—					
(1) Before treatment, . .	0·041	0·110	0·373	1·021	3·333
(2) After ,, . .	·010	·068	·277	·860	3·020
Loss of cyanide (lbs. per ton), .	·34	·48	1·1	1·9	3·6
Extraction, per cent., . .	83	84	83	85	83

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* *Proc. Chem. and Met. Soc. of S. Africa*, vol. i., 1896, p. 293.

The stronger solutions may have dissolved the gold much more rapidly than the weaker ones did although the final results were the same. The conditions under which the experiments were carried out prevented this point from being determined. Since, however, it takes two or three days in practice to wash out soluble salts from Transvaal tailings, there is little or no real advantage to be gained by dissolving the gold quickly. Crosse also found* that almost equally good results were obtained by treating rich Bonanza tailings with solutions containing from 0·1 to 0·5 per cent. KCy, the time of treatment being twenty-four hours.

Preparation and Storage of Cyanide Solution.—The cyanide is usually dissolved in a little water before being added to the stock solution, as the amount of KCy present is more easily determined in a strong solution than in any other form. A special dissolving vat of small size is often provided and is placed at a higher level than the large vats used for the storage of the stock solution. High grade cyanide is now largely used containing 98 to 99 per cent. of KCy. Sodium cyanide is also used, and this, calculated as potassium cyanide, contains 125 to 130 per cent. of available KCy. The contents of these boxes are broken into lumps and placed in wire-gauze trays, which are immersed in water contained in the stock vat. The soluble salts are soon dissolved, potassium cyanide in particular being very rapidly removed, and the tray is then lifted out, still containing the insoluble material, which is thrown away. The strength of the concentrated solution, which is kept to raise the stock solution to its normal strength, may be from 10 to 25 per cent. It should not be prepared until it is wanted, as it undergoes somewhat rapid decomposition.

Sumps.—The solutions, after passing the zinc boxes, are stored in sumps, which are now usually large steel tanks. These may be placed either at a lower or a higher level than the leaching vats. The former is the more convenient plan. A single storage tank to hold enough solution for one day's work is then placed above the leaching tanks, and is filled by pumping from the sumps once a day. In the alternative, a single sump of moderate size receives the liquid from the boxes, and as soon as it is nearly full its contents must be pumped to the large vat above the leaching tubs.

Disposal of the Tailings.—These are sometimes sluiced out of the tanks by water under pressure. The sluice gates may be of about 1 or 2 square feet in area, and are placed so that their lower edges are level with the surface of the filter. The gates are furnished with a rubber ring to be pressed tightly against a flange by a screw. Where the supply of water is not large, or the fall of the ground is insufficient for sluicing out the tailings, they are removed by shovelling out. If the vats are deep they are furnished with doors in the bottom, which are usually circular. They are often fitted with removable cylinders, 3 or 4 feet long, inside the vats, through which the upper part of the charge is shovelled. Bottom discharge doors are usually about 9 to 12 inches in diameter. A truck below receives the residues.

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* *Ibid.*, p. 326.

Samples of the residues are taken from the charge in the vat or from the ore-trucks, usually by means of a long iron semi-circular probe, shaped like a cheese-taster, which is thrust to the bottom of the vat, then revolved by means of the handle, and withdrawn with the tailings adhering to it. If the results are high, samples from various parts of the vat should be assayed separately, so as to find out if the percolation is uneven.

At the Langlaagte Estate, the five large 450-ton vats, whose upper edges are level with the surface of the ground, are discharged by a steam crane by which truck loads of tailings are elevated and placed on a tram line. The trucks are lowered into the vats and loaded up by Kaffirs, and the total cost of discharge is said to be 2d. per ton, including maintenance. The vats and sumps are excavated in the solid ground, the walls being built of brick and lined with cement.

3. Precipitation of the Gold.—The precipitation of the gold is effected by shavings of zinc freshly turned on a lathe. The shavings form a light spongy mass easily traversed by the solution and presenting a large surface for precipitation. The shavings are about $\frac{1}{500}$ inch in thickness and of no great width, but are sometimes 2 or 3 feet long. They weigh about 5 or 6 lbs. per cubic foot in the boxes, and are so fine that they can be ignited with a match, burning to zinc oxide. The shavings are placed in wooden or steel troughs, the "zinc-boxes," which are divided into compartments by partitions. They are supplied with baffle boards, which cause the solution to flow upward through the shavings. If the solution were to flow down, the gold slimes would collect on the upper surface of the zinc, and impede further flow, but by upward flowing the gold slimes are precipitated on the under surface of the zinc, from which they continually drop off, and so permit free passage of the solution. Nevertheless, Argall uses boxes in which the solution flows alternately downward and upward through the zinc, entering the first compartment at the top. Fig. 72 is from a working drawing of the steel zinc extractor boxes formerly employed at the Metallic Extraction Co.'s Mill, near Florence, Colorado. In the form now used, the double boxes are replaced by single ones 3 feet wide, with the drainage channel in the centre instead of at the sides.

The shavings are supported on wire-screen trays, formed of $\frac{1}{8}$-inch or $\frac{1}{4}$-inch iron wire gauze, so that the finely-divided, precipitated gold falls through to the bottom of the trough, while

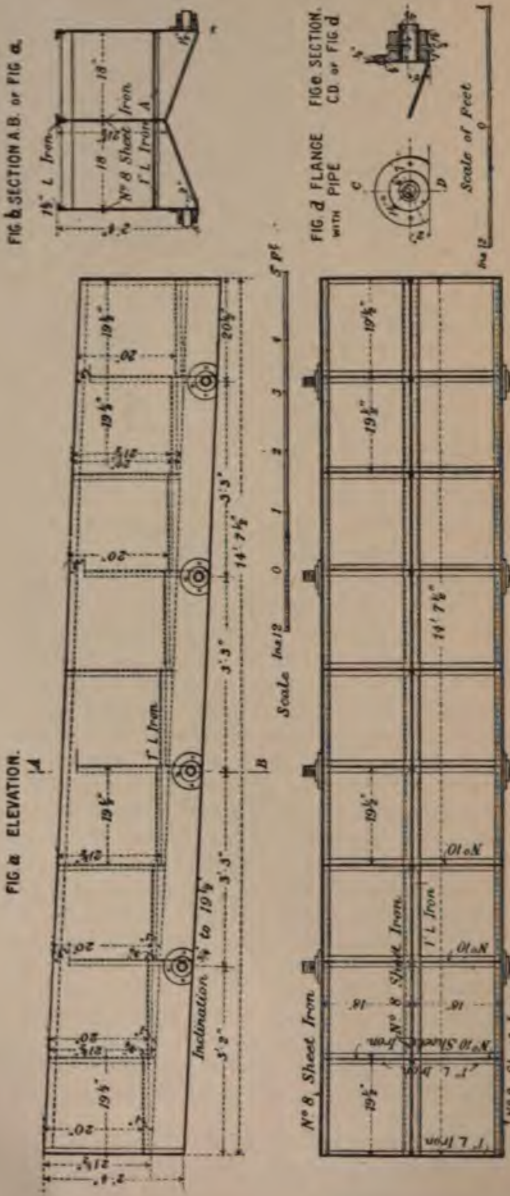
FIG b SECTION A.B. or FIG a.
1½" L Iron
N° 8 Sheet Iron
1" L Iron
FIG d FLANGE WITH PIPE
FIG e SECTION. C.D. or FIG d
Scale of Feet
FIG a ELEVATION.
A
B
1" L Iron
Inclination ¾ to 19¾
Scale Ins 12
5 Ft
14' 7½"
3' 3"
3' 2"
20¾"
19¾"
20"
FIG C PLAN
N° 8 Sheet Iron
N° 10 Sheet Iron
N° 10
1" L Iron
18"

Fig. 72.

the unaltered zinc remains on the sieve. The trays are removable and furnished with handles so that they can be lifted out. The bottom of each compartment slopes towards one corner where there is a plug hole through which the gold slimes are washed in cleaning-up. The zinc boxes are fitted with lids and kept locked up.

There are usually about ten compartments, of which at least the first and last are left empty or supplied with a sand filter to clarify the solution on entering and leaving the box by settling or filtering. The solutions are, however, often filtered in a separate box, as at the Glencairn Mine where cocoa-nut matting is used.* The use of iron in a box in contact with the zinc is to be avoided, as it increases the loss of zinc by dissolution. The use of iron wire trays, however, has not yet been avoided. Probably any other material would be less convenient.

The dimensions of the boxes depend on the amount of work to be done. In the treatment of 100 tons of ore per day, about 20 cubic feet of zinc shavings in each box is sufficient.

There should be three boxes in a cyanide works, one for the "strong" solution, one for the "weak" solution, and one for very dilute washings. In the two latter boxes less zinc is consumed, but the gold slimes are poorer, less gold being contained in the solution.

At the Glencairn Mine, where 17,000 tons per month were treated, there were seven extractor boxes, each 23 feet long, 3 feet wide, and 2 feet 6 inches deep internally. They each contained seven zinc compartments and two settling compartments, one at the head and the other at the foot, the latter also being used for dissolving the cyanide.† The fall of the boxes was 3½ inches in their length, and each was supplied by a 3-inch pipe.

At the Robinson Mine the boxes were 20 feet long and 2 feet square in cross-section; there were ten compartments in each, and about 40 lbs. of shavings in each compartment, except those at the head and foot, which were left empty in order to allow the slimes to settle. About 30 tons of solution passed through each box in a day of nine hours, so that every part of the solution was in contact with the zinc for about twenty-five minutes. The solution contained from 1 to 3 ozs. of gold per ton, and, after passing through the boxes, contained from ½ to 2 dwts. per ton. There were two sets of boxes, one for the "strong" solution, the other for the "weak."

When the solution comes in contact with bright zinc, the latter turns black at once, owing to the deposition on it of finely-divided gold. The zinc is gradually dissolved, and the shavings fall to pieces, these in the first compartment being consumed most rapidly. As the precipitation proceeds, the zinc is transferred from the lower compartments to the upper ones, and fresh zinc is added at the foot of the box. The consumption of zinc is generally from ½ to 1 lb. of zinc for 1 oz. of gold recovered.

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* *Metallurgical Engineering on the Rand*, John Yates, 1898, p. 31.

† *Ibid.*, p. 32.

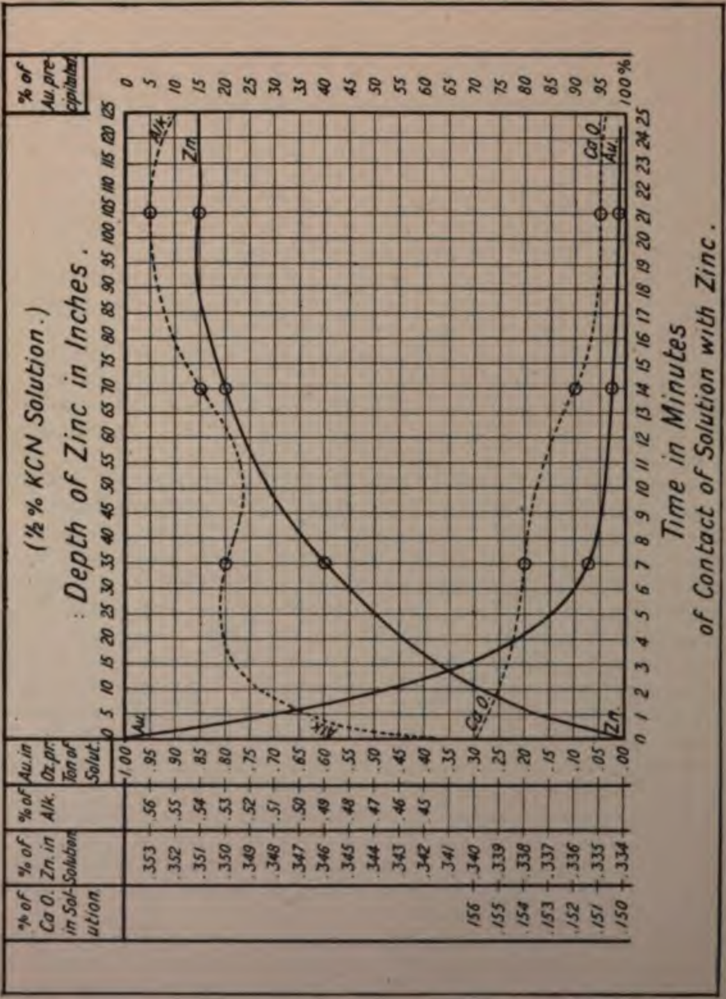

Fig. 73.

Changes in composition of the solution in passing through the zinc boxes have been investigated by P. Argall. In Fig. 73, which is kindly supplied by him, some of his results obtained in studying the conditions at the Metallic Extraction Co.'s Mill at

Florence are shown. Each double compartment contains 35 inches of zinc, and the solution was analysed when passing from each compartment to the next. The gold is seen to be precipitated mainly in the first compartment, and the amounts of alkali and zinc in solution to increase most rapidly at the same time. No reason is assigned for the shape of the curve of alkalinity, and the existence of subsidiary maxima and minima may perhaps be doubted. The cause of the precipitation of the lime is difficult to explain. It is, however, probably in existence in the solution largely as cyanide of calcium, $CaCy_2$. Exposure to air causes this to oxidise and the hydrate is then precipitated, a saturated solution in water at 15° containing only 0·13 per cent. CaO, and less at higher temperatures.

It has been proposed by Caldecott [*] to use vats instead of rectangular boxes to hold the zinc shavings, and the Cyanide Plant Supply Company manufacture enamelled iron vessels for the same purpose. The latter seem to offer great advantages in the clean-up.

Clean-up.[†]—The clean-up usually takes place either once a week or twice a month. The boxes are washed out thoroughly with clean water to remove soluble cyanides, the screens containing the undissolved shavings are lifted out and the zinc-gold slimes, remaining at the bottom of the boxes, are allowed some time to settle. The addition of alum to assist the settlement is sometimes necessary. The clear liquor is then siphoned off and the slimes allowed to run out by withdrawing a plug in the bottom of the box, and drained through a 40-mesh screen, which retains a small part only. After the residue has been rubbed on the screen with sticks tipped with indiarubber, and well washed, it is put back again into the first compartment of the box, on the top of fresh shavings, as it consists mainly of small pieces of unconsumed zinc. The subsequent fineness of the bullion greatly depends on the care with which these operations are conducted. By careful manipulation most of the undissolved zinc can be separated in this way. The slimes proper which have passed through the screen are caught in a filter-bag placed below it, squeezed as dry as possible, and are either charged wet into vats for treatment with sulphuric acid, or are carefully and slowly dried in enamelled iron pans or iron plates. Sometimes the slimes are separated and washed in a filter-press.

The richness of the dried precipitate depends on the strength of the cyanide solution, and on the time of contact as well as on the quantities of base metals which are present in the solution. By the prolonged action of the cyanide, the zinc shavings become partly corroded and disintegrated, so that the precipitated gold is mixed with zinc debris. Ordinary commercial zinc contains a considerable proportion, generally over 1 per cent., of lead, and a small quantity of carbon, besides other impurities, such as arsenic and antimony. Since the introduction of the lead-zinc couple, the proportion of lead has risen to about 5 per cent. All these accumulate and are collected with the gold.

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[*] *Chem. and Met. Soc. of S.A.*, Aug. 1899, p. 161.

[†] Alfred James, *Proc. Inst. Mng. and Met.*, vol. iii. (1895), pp. 403-404.

According to Messrs. Butters and Clennel,* the pans in use at the Robinson Works contained about 5 or 6 gallons of dried precipitate (often called "gold slimes"), which might contain as much as 150 ozs., or as little as 20 ozs., of gold. A little silver was also contained in it, the remainder being chiefly zinc and lead, with smaller quantities of tin, antimony, organic matter, &c. The average in South Africa is approximately as follows:—

Gold and silver, ...	10 to 20 per cent.
Zinc, ...	30 to 60 ,,
Base metals, silica, alumina, oxides of iron, &c., ...	20 to 40 ,,

The Hanauer Smelting Company found the precipitate from the Mercur Mine to contain the following substances:—†

Zinc, ...	39·1 per cent.
Carbonate of lime, ...	36·7 ,,
Gold, ...	4·4 ,,
Cyanogen, ...	3·5 ,,
Sulphur, ...	2·6 ,,
Iron, ...	2·4 ,,
Residue, ...	6·0 ,,
	94·7 ,,

The carbonate of lime present is probably deposited in the boxes chiefly from suspension in the solution, the gangue of the ore mainly consisting of limestone.

4. Production of Bullion from the Precipitate.—Various methods have been suggested for effecting the elimination of the zinc and other base metals. The chief ones are—

(*a*) Direct fusion with fluxes.
(*b*) Roasting, followed by fusion.
(*c*) Treatment with sulphuric acid, followed by fusion.
(*d*) Lead fusion and cupellation (Tavener process).
(*e*) Volatilisation of the zinc, followed by fusion.

(*a*) *Direct Fusion.*—This method was used in the early days of the cyanide process, but is now superseded, except perhaps on certain isolated mines. If the zinc has been imperfectly separated from the slimes, considerable quantities of nitre must be added. Clay pots are used, and in addition to nitre, other fluxes are added, such as borax, carbonate of soda, sand, and fluorspar.

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* *Eng. and Mng. Journ.*, Oct. 15, 1892, p. 365.
† *Ibid.*, Nov. 5, 1892, p. 440.

The slag, which consists of silicates of zinc, soda, &c., corrodes the pots rapidly. Large quantities of zinc oxide are given off as fumes, forming thick crusts in the flues, and evil-smelling products of decomposition of the cyanides are also evolved. The bullion produced by this method varies in colour from a pale yellow to a brownish linnet-green, and is about 650 fine, but cannot be obtained uniform in composition, so that accurate assays are difficult to obtain.

The results of analyses made on three ingots of bullion produced in this way in South Africa, and shipped to London, are appended :—

	I.	II.	III.
Gold, ...	60·3	61·7	72·6
Silver, ...	7·3	8·1	9·2
Zinc, ...	15·0	9·5	7·1
Lead, ...	7·0	16·4	4·9
Copper, ...	6·5	4·0	4·8
Iron, ...	2·2	0·3	1·4
Nickel, ...	2·0	...	...
	100·0	100·0	100·0

The slags obtained in this way are always rich in gold, part of which is sometimes in the form of shots, and may be recovered by crushing and panning. The crushed slag is then fused again with the addition of granulated lead, or of litharge, when all the gold is concentrated in the lead. If the lead thus obtained is granulated, it can be used again until rich enough to be worth cupelling.

When by the careful use of a 40-mesh screen, followed by the squeezing of the slimes in a filter bag, the zinc has been almost all eliminated by mechanical means, very little nitre is required. According to James* the zinc can be reduced to about 1 per cent., and bullion obtained 960 fine in gold and silver without the use of nitre, roasting, or acid. The dry slimes are fused in graphite pots with about half their weight of borax, a little soda carbonate, and sand or fluorspar, if necessary. The proportion of the fluxes must, however, be determined in each case by experiment. It is necessary to form a fluid slag.

(*b*) *Roasting followed by Fusion.*—The dried slimes are spread in a thin layer on iron trays, and heated to a barely-perceptible red heat in the flue of a furnace, or above a special grate under a hood. The carbon, zinc, arsenic, &c., ignite readily, being in a very fine state of division, and roasting proceeds regularly without much stirring, which tends to cause loss through dusting. Dense fumes of zinc oxide are given off, and the residue consists chiefly of the oxides of lead and zinc with gold and silver and a variable quantity of sand. About 30 per cent. of nitre is usually mixed with the wet slimes to aid in the oxidation of the base metals. Fusion with fluxes is carried out as in the previous case. The bullion produced is said to be about 800 fine. The method, like the ore previously described, appears to have been discarded, owing to the loss by dusting, which was always an unknown quantity.

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* *Trans. Inst. of Mining and Metallurgy*, vol. iii. (1895), p. 404.

(*c*) *Acid Treatment.*—The washed and filtered slimes are passed without being dried to the acid vat. Sometimes, as at Brodie,* the slimes are transferred direct into the acid vat without filtering and washing them, and are there allowed to settle partially. The supernatant liquid is then siphoned off into a settling tank. Acid is then added, a little at a time, either in concentrated or dilute form. Violent action ensues, and the fumes are highly poisonous, sometimes containing arseniuretted hydrogen, and must be carried off by a good draught. The effervescence is at first great and the vat must be deep to prevent frothing over. The charge is sometimes heated by steam, and must be occasionally stirred. The vat may be made of wood, lined with lead, or of enamelled iron.

J. E. Thomas and G. W. Williams have shown † that bisulphate of sodium is a cheaper and more convenient solvent for zinc than sulphuric acid is. In a large scale experiment at the Simmer & Jack Mill, a solution of about 34 per cent. of commercial bisulphate, equivalent to a liquid containing 9 per cent. of sulphuric acid, was used in cleaning up with satisfactory results. No arseniuretted hydrogen was produced, and there was no tendency to boil over. The residual solutions contained no free acid.

After treatment with acid the slimes are settled, washed with hot water, and separated by a vacuum filter or a filter press. They are then dried, calcined and fused in the usual way. The cost of recovery per ounce of gold at the Standard Co.'s works at Brodie, in California, is as follows:—‡

	Cost of Acid Treatment.	Cost of Fluxing and Melting.	Fineness of Bullion. Gold and Silver.
Plant No. 1, .	10·04 cents.	11·04 cents.	927·9
Plant No. 2, .	11·63 ,,	15·58 ,,	904·6

The following analyses of slimes, after treatment with acid in this way at the Brodie works, Cripple Creek, Colorado, are given by W. R. Ingalls.*

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* *Practical Notes on the Cyanide Process*, Bosqui, p. 113.

† *Journ. Chem. Met. and Mng. Soc. of S.A.*, June, 1905, p. 334.

‡ *Practical Notes on the Cyanide Process*, Bosqui, pp. 116, 117, and 118.

	I.	II.	III.
Gold,	49·85	26·18	23·60
Silver,	9·54	6·84	6·00
Silica,	15·60	10·00	6·60
Lead,	3·00	Trace	Trace
Copper oxide,	5·58	8·11	6·40
Zinc oxide,	0·14	31·80	41·50
Lime,	0·51	5·32	0·03
Ferric sesquioxide, Alumina,	4·00	6·80	9·26
Sulphur trioxide,	11·32	4·80	6·45
	99·54	99·85	99·84

In Nos. II. and III. the treatment with sulphuric acid was very incomplete. Ingalls states that in fusing these slimes No. I. would require about 50 per cent. of soda carbonate and 25 per cent. of borax, but silica in addition would be required by the others.

The following analyses show the composition of acid-treated and calcined precipitate obtained on the Rand:—

	IV.	V.	VI.	VII.
Au,	34·50	34·0	36·07	37·3
Ag,	4·75	4·3	3·5	3·7
Zn,	...	...	7·50	6·16
ZnO,	7·00	22·5	...	...
Pb,	12·50	...	20·81	20·00
PbO,	...	15·6	...	...
Cu,	2·55	...	2·46	2·5
CuO,	...	0·7	...	...
Fe,	...	...	0·02	Trace
Fe_2O_3,	3·65	2·9	...	...
Ni,	...	...	Trace	Trace
NiO,	1·00	0·7	...	...
SO_3,	6·95	6·8	6·3	13·7
SiO_2,	21·00	7·5	12·6	16·2
	93·90	95·0	89·26	99·56

In No. IV., made by A. Whitby,† the organic matter, including insoluble cyanide compounds, was not determined. In No. V.,‡ given by W. A. Caldecott, some lead existed as sulphate. The slimes were from the Robinson Deep Mill. Nos. VI. and VII., made by G. W. Williams,* are of slimes from the East Rand Proprietary. No. VI. is before and No. VII. is after calcination. No. VI. also contained from 7 to 10 per cent. of ferrocyanides (K_4FeCy_6). About half the sulphur was present as PbS, which was oxidised in the ensuing calcination.

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* *Mineral Industry for 1895*, p. 336.

† *Journ. Chem. Met. and Mng. Soc. of S.A.*, July, 1902, p. 21.

‡ *Trans. Inst. Mng. and Met.*, vol. xiv., April 13, 1905.

The charge is melted in graphite crucibles, and poured into conical moulds. The slag is fluid and glassy and contains from 20 to 200 ozs. of gold per ton, which is recovered by fusion with litharge. The bullion prepared in this way may be as high as 985 fine in gold and silver,† and should not be lower than 850 fine. It is often brittle from the presence of zinc. The loss in roasting and subsequent handling is estimated by Bettel to be from 0·01 to 0·025 per cent., or less than ¼d. per oz.

W. Bettel describes‡ a small reverberatory furnace for melting gold slimes, which he considers cheaper and better than crucibles. It resembles an English cupellation furnace,§ but has a rectangular instead of an oval opening through which can be removed a brick-lined cast-iron pan with tap-hole and spout forming the bed of the furnace. The pan is supported on two iron bearers built into the brickwork. The calcined slimes and fluxes are mixed, moistened, and compressed before being charged in. The loss by dusting is stated at only 0·04 per cent. The slag melts and is rabbled with an iron tool and tapped off, leaving the gold unmelted, and a fresh charge put in. After several charges have been treated, the heat is raised, and the gold melted and tapped out.

The method of cleaning-up in use at the Princess Works, Transvaal, is described by E. H. Johnson, as follows‖:—The slimes are transferred direct in buckets from the zinc boxes to a filter vat, 6 feet in diameter, and filtered by a vacuum-pump through closely-woven canvas, the clear liquid being returned to the boxes. Water is then added and pumped through until all soluble cyanides have been removed, and the slimes are then baled out (being weighed in the buckets during the process) into a sheet-iron tray. A closed vat, fitted with a stirring apparatus, is then charged with dilute (1 in 10) sulphuric acid, one pound of concentrated acid being added for each pound of moist slimes, the agitator is started, and the slimes added little by little through a hopper, the fumes being carried away by a 3-inch pipe. After all the slimes have been charged in, the agitation is continued for half an hour, and the hopper and sides are then washed down, the agitator washed and removed, and the vat filled with water and allowed to settle. No heat is used, except that caused by the mixing of strong sulphuric acid with water in the vat and by the action on the slimes. The clear liquid is siphoned off, and washing by decantation repeated four or five times, the washings containing about 13 grains of gold per ton of solution. The gold residues are then dried in enamelled iron dishes, roasted gently in sheet-iron trays, and then ground, mixed with fluxes, and transferred to the crucible. The charge fuses quietly with little fume, yielding bullion amounting to 50 to 60 per cent. of the weight of the slime. The average fineness in 1896 was 821·9, and the slags assay 23 ozs. per ton, but only 0·2 per cent. of the gold is locked up in this way. The cost of reduction in an actual case was as follows:—

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* *Trans. Inst. Mng. and Met.*, vol. xiv., April 13, 1905.

† *Loc. cit.*

‡ *Chem. and Met. Soc. of S.A.*, Feb. 1899, p. 41.

§ See description of this in Collins' *Metallurgy of Silver*, 1899, p. 339.

‖ *Chem. and Met. Soc. of S. Africa*, June 19, 1897.

Dry weight of zinc-gold slimes,	504 lbs.		
,, after acid treatment,	100 ,,		
672 lbs. of acid at 4½d.,	£12	12	0
66 ,, borax at 37s. 6d. per cwt.,	1	2	1
9 ,, soda carbonate at 2¼d.,	0	1	10
9 ,, fluorspar at 4d.,	0	3	0
5 bags coke at 8s. 6d.,	2	2	6
1 No. 60 crucible,	1	7	6
	£17	8	11

Yield 620 ozs. fine gold. Cost 6·7d. per fine oz., without cost of labour.

When a filter-press is used, less gold is lost in the washing, and by the use of hot water, the zinc sulphate is more perfectly removed, so that the subsequent smelting is more easily and cheaply executed.*

The composition of acid-treated precipitate in the Transvaal is given above, p. 301. It was formerly customary to fuse such material with nitre, sand, and borax, but in 1902 Johnson and Caldecott † described the use of manganese dioxide instead of nitre. Fluxes used recently are as follows: ‡—

Acid-treated dried slimes,	100 parts.
Borax,	45 ,,
Soda carbonate,	1 ,,
Fluorspar,	7 ,,
Manganese dioxide,	2 ,,
Sand,	15 ,,

This gave bullion from 875 to 900 fine.

(*d*) *Tavener Process.*§—In this process the zinc-box precipitate is filter-pressed, the fine zinc having previously been separated. The slimes are partly dried, rubbed through a coarse sieve, mixed with fluxes, again sieved, and shovelled into a reverberatory furnace. The fine zinc is also mixed with fluxes, and charged in on the top of the slimes. A cover of litharge and old slag is then added, and the charge is ready for smelting. The proportions given by P. S. Tavener were as follows :—

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* *Trans. Inst. Mng. and Met.*, vol. v. (1897), p. 147.

† *Journ. Chem. and Met. Soc. of S. Africa*, vol. iii. (1902), p. 21.

‡ E. E. Meyer "On the Fluxing of Gold Slimes," *Journ. Chem. Met. and Mng. Soc. of S. Africa*, vol. v. (1905), p. 169.

§ *Journ. Chem. and Met. Soc. of S.A.*, Oct. 1902, p. 70.

A.	Slime, .	. 100	parts by weight.	B.	Fine zinc, .	. 100	parts.
	Litharge,	. 60	,, ,,		Litharge, .	. 150	,,
	Assay slag,	. 10-15	,, ,,		Slag, . .	. 20	,,
	Old slag,	. 10-15	,, ,,				
	Silica, .	. 5-10	,, ,,				
	Sawdust,	. 0·6-1·2	,, ,,				

The fire is then lighted, and the charge slowly dried and melted. Very little zinc is volatilised. The slag is stirred and more sawdust added, and when the slag is judged to be free from litharge, it is run off into slag-pots. The last of the slag is removed by skimming. The lead is then stirred, sampled, and drawn off into moulds. The usual charge yields about 12,000 ozs. of soft lead, containing about 8 per cent. of gold.

In the second operation the lead is cupelled in an English cupellation furnace. It would probably be better to make two operations of this, raising the fineness to 50 or 60 per cent. of gold and silver in one furnace and finishing in another, but Tavener does not propose this. The result is that the rich litharge flowing off at the end of cupellation is not kept separate. It is also stated that it is difficult to prevent the charge from freezing when most of the lead has been removed. The cupelled gold is allowed to set on the test, and is then prised off, broken up while hot, and melted in crucibles. Tavener states that the gold and silver carried off by the litharge is not lost, as the litharge is used over again. There is a limit to this system, as the copper would not be slagged off in the reverberatory furnace and would accumulate in the litharge, which would eventually become too impure for further use.

The losses in the process are unknown, but are believed to be far less than in the methods previously described. The cost of treatment is stated by Tavener to be about 3d. per oz. of gold in South Africa, as against about 1s. per oz. for the acid treatment, calcining, and smelting process.

(*e*) *Volatilisation.*—It was found by Sulman & Picard * at Deloro that great losses of gold were experienced in driving off the zinc by volatilisation. These losses were mechanical, due to the bursting of the bubbles while the zinc was boiling. The loss was reduced to less than 0·1 per cent. by briquetting the slimes with molasses before driving off the zinc. The zinc is condensed and saved. The retort residues are smelted.

Treatment of Ore Slimes by Decantation.—On the Rand, there had accumulated by the year 1897 hundreds of thousands of tons of slimes which could not be treated until then, owing to their impermeability, although the gold contained in fresh slimes is readily soluble in the laboratory. According to Dr. Loewy * the Rand slimes contain over 70 per cent. of silica, 16 per cent. of alumina, 4 or 5 per cent. of iron sesquioxide, 1·25 per cent. of magnesia, and about 3 per cent. of combined water. There is thus much free silica present in them, and only 35 or 40 per cent. of hydrated silicate of alumina or clay. The average value of the Robinson slimes was between 7 and 8 dwts. of gold per ton, and the fineness was such that it would pass through a 225-mesh screen. It was proposed by Bettel to treat this material in Johnson's filter-presses. The slimes were mixed with a very dilute (0·01 per cent.) cyanide solution and thoroughly agitated, and then filtered, and water forced through under a pressure of 100 lbs. to the square inch. It was stated that about 98 per cent. of the assay value of the slimes could be extracted in this way, but the method was never applied on the large scale in Africa.

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* *Journ. Soc. Chem. Ind.*, Dec. 31, 1897, p. 961. *Mineral Industry for 1897*, p. 347.

The following method of treating slimes, devised by J. R. Williams,† was started at the Crown Reef Gold Mining Company's Mill in August, 1896, and worked satisfactorily:—To the water carrying the slimes from the separating plant, milk of lime is added, and the slimes thus precipitated in a flocculent form. An automatic lime feeder is used, as too much lime is as bad as too little (a large excess of lime preventing the subsequent precipitation of the gold), and regularity of feeding is therefore essential. Three large pointed boxes, two of them 20 feet × 20 feet and 10 feet deep, and the third 40 feet × 40 feet and 10 feet deep, are used for settling the slimes, which are then drawn off at the bottom and pumped into the first two of the eight treatment vats, about 90 per cent. of the water contained in them having been separated. The vats are each 32 feet in diameter and 10 feet deep, with conical bottoms. More water having been separated from the slimes by allowing them to settle, they are sluiced into a pump by a jet of cyanide solution, and transferred to the second series of two vats. These vats are filled with solution, the strength of which is brought up to 0·01 per cent. KCy. About 80 per cent. of the gold is dissolved in the passage through the pumps, but agitation is continued for from one to two hours by withdrawing the solution at the bottom and discharging it in oblique jets at the tops and through the sides. The slimes are then allowed to settle, and the clear solution drawn off through side-cocks (which have been advantageously replaced at the Rand Central Works by syphon pipes) and passed to the precipitation boxes. The slimes are then pumped successively into the third and fourth series of two vats, where they

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* *Chem. and Met. Soc. of S. Africa*, June, 1898, p. 80.

† *Ibid.*, July 17, 1897.

are further agitated with very dilute solutions of cyanide, and allowed to settle. These solutions are not passed to the precipitation boxes, but are transferred to the preceding series of vats, those going to the second series having cyanide added to them to bring up their strength.

The "strong" solution from the second series of vats is run into two settling tanks 15 feet in diameter and 5 feet deep, where it is allowed to clarify, the tanks being used alternately (as in the case of each of the other sets of vats), one being filled up, while the liquid from the other is clarifying by settlement and passing to the precipitation plant where 8 tons of solution per hour were treated. Electrical precipitation is used, and there are four boxes, each 30 feet × 6 feet and 4 feet 9 inches deep. The iron anodes are set at 6-inch centres, and between each pair is placed a double wire frame, carrying on each wire three lead sheets cut into strips. The solution, which contains 60 grains of gold per ton, passes upwards through these, and thence to the sump for use over again. The liquid takes about 12 to 14 hours to pass through the boxes, and the rate of precipitation to be expected may be judged from the following experimental trial :—

No. of Test.	Time of Precipitation.	Assay of Liquid.
		Per Ton.
1	Before treatment.	2 dwt. 12 gr.
2	After 3 hours.	1 ,, 4 ,,
3	,, 6 ,,	0 ,, 20 ,,
4	,, 9 ,,	0 ,, 12 ,,
5	,, 12 ,,	0 ,, 8 ,,
6	,, 15 ,,	0 ,, 4 ,,
7	,, 18 ,,	0 ,, 2 ,,
8	,, 21 ,,	0 ,, ½ ,,
9	,, 24 ,,	Trace.

About 90 per cent. of the gold is precipitated in practice, according to this table.

The cost of treatment for the months of April and May, 1897, was as follows :—

Management, assaying, and labour,	13·99d. per ton.
Power,	2·75d. ,,
Cyanide,	5·51d. ,,
Lime,	6·19d. ,,
Lead foil,	1·75d. ,,
Sundries,	1·92d. ,,
Royalty,	4·77d. ,,
Maintenance,	8·17d. ,,
Total,	3s. 9·05d. ,,

The results obtained were as follows :—Tons treated (?), 6,643; assay value before treatment, 5·32 dwts., after treatment, 0·892 dwt.; theoretical extraction, 83·6 per cent.; fine gold recovered, 1067·16 ozs.; actual extraction, 60·5 per cent.; profit, £3,058 11s. 2d.

The difference between the theoretical and actual extraction is partly due to the great difficulty in estimating the tonnage of slimes treated. The clean-up takes place every two months. The amount of lime used is large, and is in proportion to the amount of water and not to that of slimes, so that if, as would be the case at some mills, there was double the quantity of slimes in the water, the same amount of lime would precipitate them.

The description of the decantation method given above is an abstract of that originally given by J. R. Williams. A few additional details are appended, but the practice has not been materially altered. The amount of lime added to the pulp is about ½ lb. per ton of pulp or less. Caldecott states [*] that with properly adjusted spitzkasten, 0·003 per cent. of alkali (equal to 0·0021 per cent. CaO) is sufficient to ensure settlement. This is only about 1 oz. of lime per ton of pulp. The amount usually added, however, is from 2 to 20 ozs. per ton of pulp, which may amount to from 3 to 10 lbs. of lime per ton of dry slimes. The slimes are agglomerated into larger particles, and then rapidly settle. The effect is produced by dissolved lime, not undissolved. Many other substances also cause agglomeration, such as ferrous sulphate and other sulphates, acids, &c.

It is now usually the practice to add lime to the ore as it is charged into the batteries. The alkalinity of the water is thus made to equal from 0·005 to 0·05 per cent. The first additions of lime to the stamp batteries of the Rand made the plates dirty and the results of amalgamation less satisfactory, but subsequently they improved. The beneficial effects of the lime are no doubt due in part to the agglomeration of the slimes, by which better contact between the finest material and the plates is obtained. Lime would also remove grease and decompose the ammonia salts of organic matter, and so improve the amalgamation in that way.[†]

The agglomeration of the battery pulp also leads to a larger percentage of material being caught in the sand separators, so that the slimes are reduced in quantity and also impoverished by this method of treatment. At the Gledenhuis Deep Mill, the results of the change in practice were given by J. R. Williams as follows [‡]:—

Year.		Extraction by amalgamation.	Assay value of slimes.
1897,	No lime in battery, . .	49·63 per cent.	4·21 dwts.
1898,	Lime added in battery, .	54·53 ,,	2·93 ,,

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[*] *Chem. and Met. Soc. of S. Africa*, July, 1898, p. 96.

[†] Pauli and L. Ehrmann, *Ibid.*, July, 1899.

[‡] *Journ. Chem. and Met. Soc. of S.A.*, April, 1899, p. 97.

At the New Heriot Mill, the proportion of slimes was reduced by the addition of lime from 23·57 to 19·74 per cent. of the ore, and the assay value was reduced from 3·13 to 2·38 dwts.*

The surprising result of the attempt to treat the slimes was, therefore, to reduce their value, and in certain cases to make it unprofitable to treat them.

The slimes pulp flowing from the sand separators usually contains about 5 per cent. of dry slimes. This is raised to 10 per cent. by treatment in spitzkasten, and to 50 per cent. by settling in the treatment vats, the clear water being returned to the mill. Large dams for the settlement of slimes are thus rendered unnecessary, and the consequent loss of water by evaporation avoided. Thus at the Treasury Mine, according to Spencer,† the monthly consumption of water before the slimes plant was erected was 5,038,000 gallons, and after it was brought into use only 2,533,000 gallons.

In settling the slimes, the clear supernatant liquid is drawn off by a pipe working on a swivel joint, and having its aperture held at the surface of the liquid by a floating ball. Four or five tons of cyanide solution are added per ton of settled slimes, and after agitation and settlement this is repeated, over 95 per cent. of the dissolved gold being thus removed. In practice, less solution than this is often used. A valve for admitting air to the circulating pump ensures the efficient aëration of the pulp. Mechanical stirrers are never now used. The residues after the final settlement are generally sluiced into a pump and discharged into a launder, down which they run to the dump.

The precipitation of gold from the very dilute solutions obtained in slimes plants is effected either by the zinc-lead or the electrical process, both of which are described subsequently. The cost of treatment at the Crown Deep Slimes plant during three months in 1898 was 2s. 9d. per ton, 11,280 tons being treated, and 87·49 per cent of the gold extracted. The slimes contained 3·157 dwts. per ton, and the amount of solution passed through the extractor boxes was 21,842 tons.‡

The estimation of the weight of dry slimes per ton of pulp is made as follows:—The density of the dried slimes is determined by the specific gravity bottle, with the usual precautions.

Then if

w = weight of 100 c.c. of slimes pulp in grammes,
d = density of the dried slimes,
x = weight of dry slimes in 100 c.c. of pulp in grammes,

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* Dessauer, *S. A. Mining Journal*, April 29, 1899.

† See paper by J. R. Williams, *Chem. and Met. Soc. of S. A.*, April, 1899.

‡ T. L. Carter, *Chem. and Met. Soc. of S. A.*, 1898, p. 143.

It follows that

$$d = \frac{x}{x + 100 - w}$$

$$\text{and } x = \frac{d\,(w - 100)}{d - 1}.$$

Then the percentage of slimes in the pulp is $\frac{100\,x}{w}$.

A diagrammatic plan of a plant* for the treatment of 30 tons a day of slimes is shown in Fig. 74. The vats are all of steel, 30 feet in diameter, with conical bottoms, and are 11 feet 3 inches deep in the centre, and 9 feet deep at the sides. The sumps are 25 feet in diameter. The final vat has bottom discharge. The extractor boxes are not shown.

The Treatment of Accumulated Slimes.†—Although the gold in slimes fresh from the battery is very readily soluble in cyanide solution, it is quite otherwise with slimes which have accumulated in dams and settling pits and have been exposed to the weather for some time. The presence in these materials of finely-divided ferrous sulphide, ferrous hydrate, and other ferrous salts and decomposing organic matter gives rise to the rapid abstraction of oxygen from the cyanide solutions used in their treatment, and the solution of the gold is in this way absolutely prevented. Even in fresh slimes some ferrous sulphide, produced during the crushing in the battery, may be present, as Caldecott has shown that this substance is produced by triturating pyrites in a mortar. The obvious remedy for this difficulty is to supply oxygen artificially, either in the form of air delivered into the agitation pump, or in the form of an oxidising agent, such as potassium permanganate. Aëration of the pulp in the dissolving vats was regularly used from the end of 1896 at the Rand Central Ore Reduction Co.'s Works, formerly accelerated, when much organic matter was present, by the addition of $\frac{1}{8}$ to $\frac{1}{2}$ lb. of permanganate per ton of dry slimes.

Old accumulated slimes are often highly acid, a consumption of from 8 to 20 lbs. of lime per ton being not uncommon. Ferrous hydrate, produced by the action of lime on the ferrous sulphate, either existing originally in the slimes or produced by aëration, tends to abstract oxygen from the solution and also to form ferrocyanide with KCy, and hence the ferrous hydrate is carefully oxidised by aëration to form ferric hydrate. After this is done the consumption of the cyanide in the dissolving vats is only about $\frac{1}{4}$ lb. per ton, even with the strongly-acid accumulated slimes, another ¼ lb. per ton being decomposed in the boxes or thrown away with the residues.

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* This is reproduced by the kind permission of the Cyanide Plant Supply Company.

† See W. A. Caldecott's paper in *Proc. Chem. and Met. Soc. of S. Africa*, July 17, 1897.

Treatment of Ore Slimes by Filter Pressing.—The slimes met with at Kalgoorlie are far richer than those treated on the Rand, and water is scarce and expensive. For these and other reasons the decantation method was never installed in Western Australia although experimentally successful. The oxidised slimes at first encountered were either passed direct to filter-presses, and cyanide forced through the cakes of slimes by com-

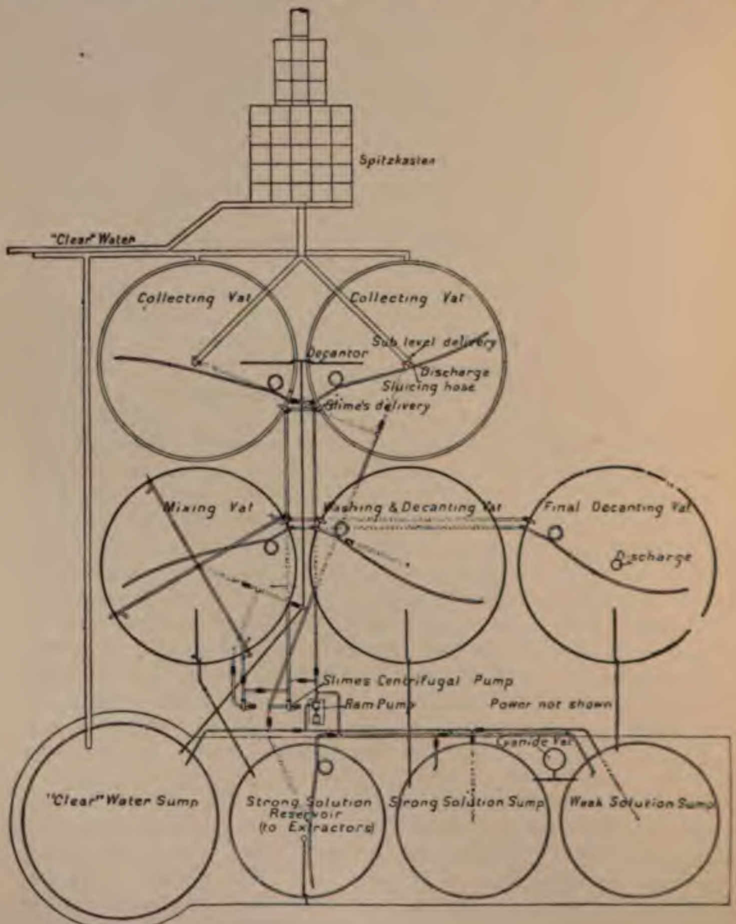
Spitzkasten
"Clear" Water
Collecting Vat
Collecting Vat
Sub level delivery
Decantor
Discharge
Sluicing hose
Slime's delivery
Mixing Vat
Washing & Decanting Vat
Final Decanting Vat
Discharge
Slimes Centrifugal Pump
Ram Pump
Power not shown
Cyanide Vat
"Clear" Water Sump
Strong Solution Reservoir (to Extractors)
Strong Solution Sump
Weak Solution Sump

Fig 74.

pressed air, or agitated with cyanide in vats, and after the gold had been dissolved, filtered, and washed in filter-presses. The rate of solution was sometimes accelerated by aëration or by the addition of bromine (Sulman-Teed process). Bromo-cyanogen is necessary when unoxidised tellurides are present, as telluride of gold is not attacked by ordinary cyanide.

The method employed for oxidised ore at Hannan's Brownhill Mine, Kalgoorlie, was briefly as follows:—*The ore is crushed dry to 25- or 30-mesh, and delivered to a dry separator where it is divided into about equal parts of slimes and sands. The Mumford and Moodie separator is shown in Fig. 75. The crushed ore is raised by the elevator, A, and falls into the hopper, C, which is always kept partly full, even distribution being aided by the rotating baffle disc, B. The disc, F, and fan, G, also rotate with the spindle. The ore falls on F and is thrown

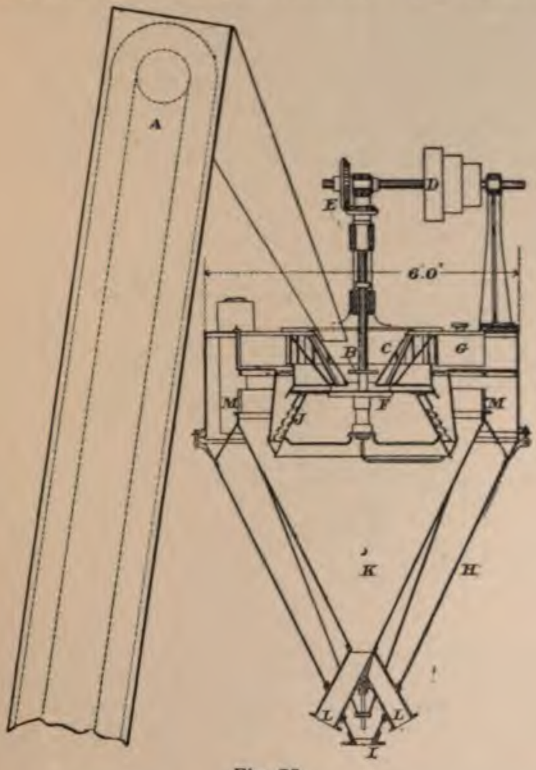

Fig. 75.

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* H. M'Neill, *Trans. Inst. of M. and M.*, vol. vi. (1898), p. 250.

off by centrifugal force, and air currents from the fans then carry off the fine dust to the outer chamber, H, where it settles and is delivered at I. The coarse particles settle in K against the air current, and are withdrawn at L.

The slimes separated in this way were fed into the vat, A (Fig. 76), and agitated by paddles with cyanide solution. When the gold had been dissolved, which occupied about three hours, the slimes were run into the montejus or air pressure receiver, C, whence they were forced by means of air at a pressure of 60 to 80 lbs. per square inch into the filter presses, D, the filtrate from which passed through the pipe, B, into the vat, E, and thence in a constant stream to the zinc boxes. The cakes were washed in position with clear water, and then the press was opened, and the cake falling into the truck, F, was trammed to the dump. In later practice the presses were filled by a pump instead of by air pressure.

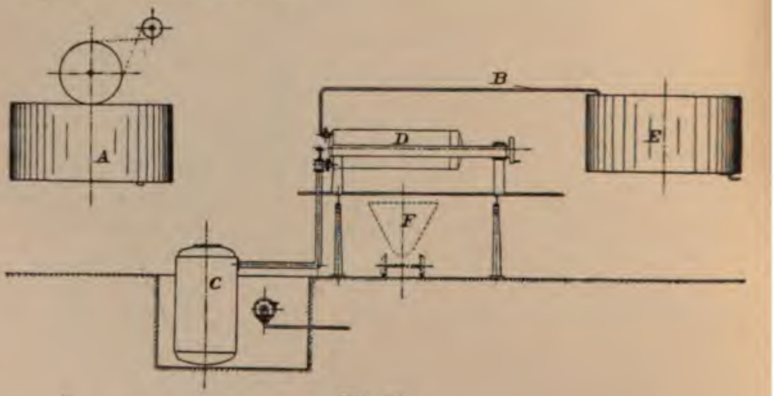
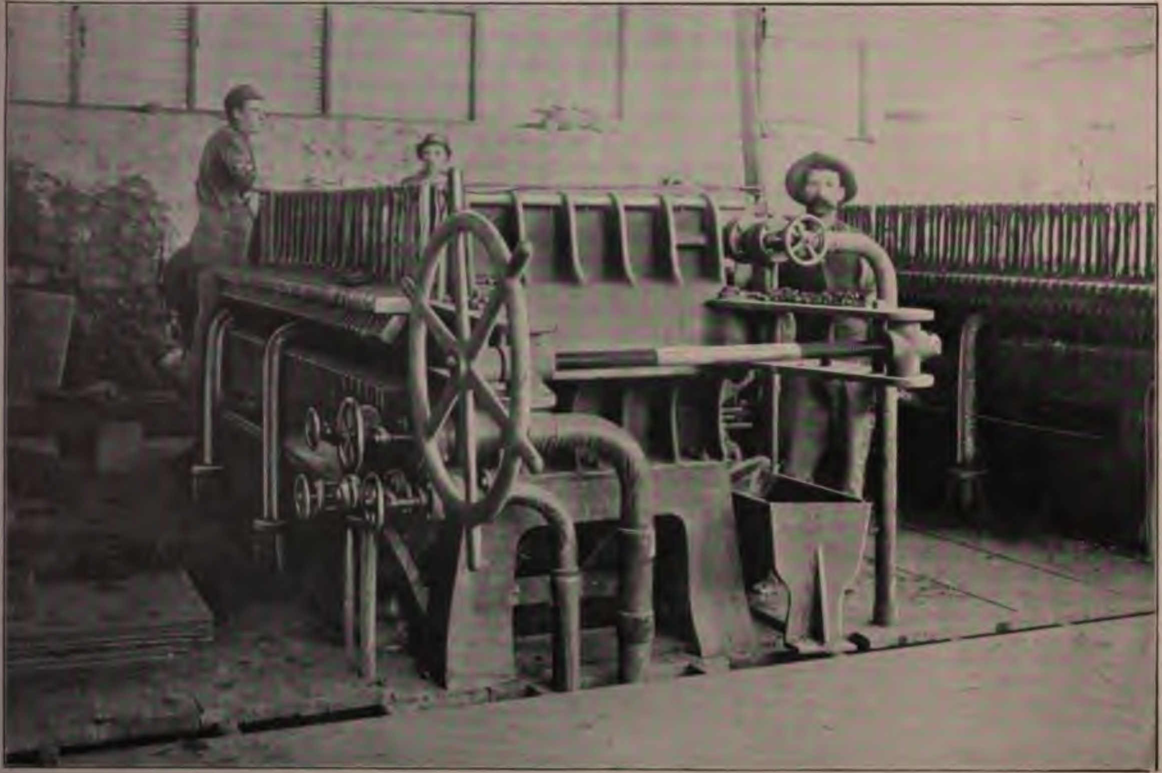


Fig. 76.

The filter presses had 20 chambers each, and formed cakes 28 inches square and 3 inches thick, the total charge being 1¼ tons of slimes. In later practice larger presses were used with 50 chambers 3 to 4 feet square. These hold 5 or 6 tons of wet slimes.

A filter press is shown in Figs. 77 and 78. A distance frame (Fig. 78 and Fig. 78*a*, E) lies between each two plates. Each plate and frame has 4 holes, A B C D, which correspond in position, so that when the plates are fitted together, the holes form passages through all the plates in the press. The slimes from the receiver pass through the hole, A, and so enter the spaces, K, inside the distance frames. When the spaces, K, are filled, the liquid begins to be forced through the filter-cloths and passes to the channels, B and C. When the cell is full of compacted nearly dry slimes, the inlet at A and the outlet at B are closed,

PLATE VI.



Filter Press, Kalgoorlie, Western Australia.

From a Photograph kindly supplied by Mr. A. G. Charleton.

and water is forced in at D and runs out at C, passing through the filter-cloths and slimes, and displacing the cyanide solution. The washing is said to be rapid and effective.

It was stated* that the slimes treated in this way consisted largely of fine sand. If this were removed the leaching was made impossible.

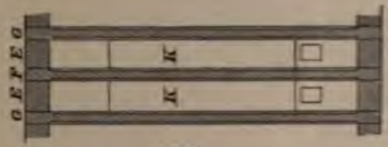


Fig. 78a.

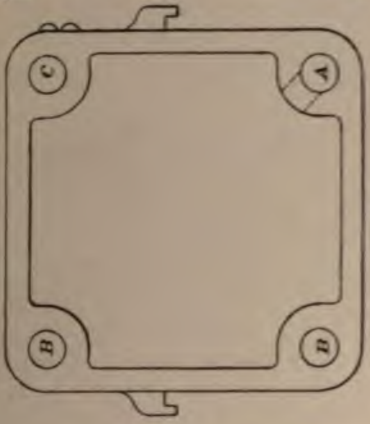


Fig. 78.

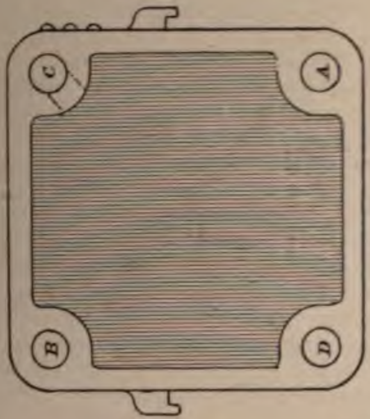


Fig. 77.

The time of treatment in the filter presses is about one hour, of which 20 minutes is occupied in discharging the presses. The cost per ton at the Lake View Consols, where a similar method of treatment was used, is given as follows by Alfred James:—*

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* H. L. Sulman, *Ibid.*, p. 257.

	s.	d.
Superintendence and settling solutions,	0	6·8
Discharging and filling presses,	0	34·4
Compressed air,	0	9·7
Turning zinc,	0	·2
General repairs,	0	3·0
Assaying, retorting, and smelting,	0	3·0
Zinc,	0	·5
Filter cloth,	0	3·1
Cyanide,	0	10·7
General stores and charges,	0	4·4
Electric light and maintenance,	0	3·5
	6	7·3

Filter-pressing has more recently been introduced into South Africa, but it has hardly yet begun to displace the decantation method.

Treatment of West Australian Sulpho-Telluride Ore.—The Kalgoorlie sulphide ores contain from 0·03 to 0·15 per cent. of tellurium, 3 to 7 per cent. of iron pyrites, 6 to 17 per cent. of calcite, about 2 per cent. of soluble salts (sodium chloride, magnesium sulphate, &c.), a large percentage of silicate of alumina, and from 0·1 to 0·3 per cent. of copper. Much of the ore is rich in gold, and enormous quantities contain more than 1 oz. per ton. After numerous experiments, two processes were devised by which treatment is fairly successful. These processes are described below.

The *Marriner process*,† in work at the Great Boulder Main Reef and the Great Boulder Proprietary Companies' Mills, consists in (1) dry crushing in Gates' rock breakers, followed by ball mills or roller mills, (2) roasting dead, (3) grinding wet in pans with a large quantity of mercury and hot alkaline cyanide solutions, (4) agitating by means of paddles in large vats with cyanide solutions containing from 0·01 to 0·08 per cent. KCy, (5) filter-pressing, and (6) precipitation of the gold by zinc.

In roasting, a large amount of soluble sulphates is formed, and the losses by dusting must be carefully attended to. Various furnaces are used (see pp. 224-230). The pans are used primarily to grind the ore to fine slimes, which is necessary to enable the cyanide to dissolve the gold, but any coarse gold is extracted in them by amalgamation. The agitators are from 10 to 21 feet in diameter and 5 to 16 feet deep, and are usually fitted with a vertical shaft revolving by bevel-gearing at the top and carrying two sets of radial arms at the bottom. The total cost of treatment in 1900 was put at 20s. per ton, including 4s. for roasting and 6s. to 8s. for filter-pressing. In 1904 it was given by Donald Clark as 16s. 11·44d. per ton. The tailings assay from 1 to 2 dwts., varying with the ore, and the cyanide consumption is from 1½ to 2 lbs. per ton.

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* *Trans. Inst. Mng. and Met.*, vol. viii. (1900), p. 515.

† James, *Trans. Inst. M. and M.*, 1900, p. 490, and Donald Clark, *Australian Mining and Metallurgy*, pp. 15-44.

The precipitated gold slimes are treated with sulphuric acid in a lead-lined cast-iron pot and filter-pressed in a press with plates of a lead-antimony alloy. The cakes are washed and dried and then smelted in a Faber du Faur zinc-distilling tilting furnace. This furnace is described in works on the metallurgy of zinc. It holds about 8 cwts. of material, and consists of a large inclined graphite pot shaped like a bottle, having a contracted mouth, and contained in a cubical tilting furnace.

In the ***Diehl process***,[*] which was introduced at the Hannan's Star Mine, and subsequently used at the Lake View Consols and at Hannan's Brownhill, the ore is wet crushed, amalgamated, concentrated, separated into sands and slimes, the sands reground in tube mills, and the whole agitated with cyanide, to which bromo-cyanide is added and filter-pressed. The concentrates, which contain 30 to 40 per cent. of the values, are roasted and sent back to the wet crushing mill. The cost has been given as 28s. 3d. per ton, and with ore assaying 16 dwts. 14 grs., the final residue contained 26 grs., or 6½ per cent.

According to Knutsen, the ore at Hannan's Brownhill is crushed in a 20-stamp battery fitted with inside and outside amalgamated plates. An average of 75 tons of ore are crushed per diem through 30-mesh screens. These are 4 Wilfley concentrators, and from 3 to 5 per cent. of concentrates are produced. These are roasted in an Edward's mechanical furnace and sent back to the stamp mill. The tailings pass through two classifiers and the separated sands are re-ground in two slime or tube mills, each containing 2½ tons of flint balls. The product of the tube mills goes back to the classifiers. The slimes are concentrated by the removal of water in spitzkasten, and the sludge, containing 40 to 45 per cent. of solid slimes, flows to one of five agitators, each 20½ feet in diameter and 7½ feet deep. When the agitator is full, it is charged with strong KCy solution, and two hours later with strong BrCy solution. The amounts are KCy 0·20 per cent. and BrCy 0·05 per cent. of the solid material. After about 24 hours' agitation, lime is added, the quantity being 3 to 4 lbs. per ton of solid slimes, and about two hours later the charge is filter-pressed. Each of the two presses holds 5 tons, and the time of treatment is about two hours. The gold solution is again filtered before flowing to the zinc boxes. The filter-press cakes, after being washed and then dried by air-blowing, go to the dump.

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[*] "The Diehl process" by H. [illegible] *Inst. M. and M.*, vol. xii. (1902), p. 1.

They assay from 1 to 2 dwts., whether the original ore contained 4 ozs. or 1 oz. per ton. The consumption of chemicals is 3 lbs. KCy and 1¼ lbs. BrCy per ton, costing about 8s. per ton.

The total cost in July 1901, was as follows:—

	Per. ton. s.	d.
Milling, ...	4	0·59
Concentration, ...	1	7·19
Treatment of concentrates, ...	1	4·08
Extraction, ...	17	1·02
	24	0·88

Electrical Precipitation.—In the Siemens-Halske process the gold is deposited from solution by the passage through the liquid of a current of electricity. Moreover, as the precipitation is as readily obtained in extremely dilute cyanide solutions as in those containing 0·1 per cent. or more, very weak solutions are used in dissolving the gold from the ores when electrical precipitation is employed. Gold can be extracted from some ores as completely by a solution containing 0·03 per cent. of cyanide as by one containing 0·3 per cent., the only difference being that the time required is sometimes considerably longer. On the other hand, the advantage in using the more dilute solution is that the selective action in favour of the gold is increased, and the amount of cyanide decomposed by "cyanicides" in the ore is diminished. In addition to this, some cyanide solution is invariably left in the ore, and if the "weak" solution used to finish the dissolution of the gold contains only 0·01 per cent. of cyanide, instead of 0·1 per cent., the amount of cyanide lost in this way by mechanical means is also reduced.

The process was first adopted on a large scale at the Worcester mill in the Transvaal. Here the vats are 20 feet in diameter and have their sides formed of staves 10 feet long; the five vats hold 135 tons each. The battery pulp, after passing over Frue vanners, is classified into four products by hydraulic classifiers. The first classifiers, which consist of spitzlutten, remove the coarse sand and pyrites, amounting to 15 per cent. of the pulp, and containing 15 dwts. of gold. This product is treated for nine days with solutions of 0·08 per cent. of cyanide, and, after being washed with 0·01 per cent. solutions, gives residues assaying 1½ to 2 dwts., so that from 87 to 90 per cent. of th gold is extracted. The second product yielded by the hydraul classifiers comprises 50 per cent. of the pulp, contains 6 dw of gold per ton, and, after five days' treatment with soluti ranging from 0·05 per cent. downwards, yields residues c taining from 1 to 1·25 dwts., showing an extraction of 80 to per cent. The finest sand is separated from the slimes pointed boxes; the slimes amounted to 25 per cent. of the w pulp, and assay 4½ dwts. The fine sands, constituting 10 cent. of the pulp, contain 4½ dwts. of gold, and after treatment yield residues assaying 1 dwt. per ton. The consumption of cyanide averages ¼ lb. per ton of the tailings, of which 3,000 tons are treated per month.

The precipitation plant consists of four boxes, each 18 feet long, 7 feet wide, and 4 feet deep. Copper wires are fixed along the tops of the sides of the boxes, and convey the electric current from the dynamo to the electrodes. The anodes are made of iron, and the cathodes, on which the gold is deposited, of lead. It was stated that amalgamated copper plates had been tried but abandoned, as the mercury penetrated the copper under the influence of the current, and a dry amalgam resulted which did not adhere to the plate. Lead answers all the requirements of the cathode laid down by von Gernet, which are—(1) that the precipitated gold must adhere to it, (2) that it must be capable of being rolled out into very thin sheets to avoid unnecessary expense, (3) that it must be easy to recover the gold from it, and (4) that it must not be electro-positive to the anode, in order to prevent return currents being generated when the depositing current is stopped. A fifth requirement might be added, that the gold should be separable from the cathode without destroying the latter. This requirement is not fulfilled by lead.

At the Worcester mill the anodes are iron plates, 7 feet long, 3 feet wide, and ⅛ inch thick; they are supported in a vertical position by wooden strips nailed to the box, and are covered with canvas to retain the small quantity of Prussian-blue produced. The sheet-lead cathodes are stretched on wires fixed in light wooden frames which are suspended between the iron plates. There are in all 3,000 square feet of cathode surface. The solution is made to circulate between the plates passing alternately over and under the edges of the anodes. The boxes are kept locked except when cleaning up is necessary, when the cathodes are removed and replaced with fresh sheets of lead. The lead, which contains from 2 to 12 per cent. of gold, is then melted into bars and cupelled. The consumption of lead is 750 lbs. per month, its cost being equal to 1½d. per ton of tailings. The gold is comparatively free from base metals. The total cost of treatment at the Worcester mill is under 3s. per ton of tailings. The electrodes are placed 1½ inches apart, and a current of 4 volts is employed, giving about 0·06 ampere per square foot. In later practice, according to Butters, the boxes are reduced in size, being now 29 feet long, 3½ feet wide, and 2 feet 9 inches deep, the iron anodes are enclosed in canvas sacking, and the space between them is filled loosely with lead turnings, which have replaced the sheet lead formerly used as cathodes. The liquid circulates through the turnings.

One of the main difficulties in electrical precipitation lies in the great resistance offered to the passage of the current by the very dilute solutions employed. In order to reduce this as far as possible, the electrodes must be of very large size, and the distance between cathodes and anodes must be small.

The ions of aurocyanide of potassium are K and $AuCy_2$. As the result of electrolysis, therefore, potassium is set free at the lead cathode, where it attacks the water, forming potash and hydrogen; at the same time the gold in the double cyanide is displaced and precipitated, both by the potassium and the nascent hydrogen. Such hydrocyanic acid as may be formed is neutralised by the potash. Meanwhile the anion, $AuCy_2$, is set free at the anode, but is at once split up into AuCy and Cy; the latter unites with the iron, forming cyanides, which become converted into Prussian blue, and are also oxidised in part, forming ferric oxide, and the cyanide of gold is partly precipitated in this substance, although if enough free potassium cyanide is present it may be kept in solution; even if it is precipitated, it can afterwards be recovered by aëration in an alkaline solution of potassium cyanide.

It is, however, clear, that the solutions must be circulated, and also that the more free cyanide is present, the greater is the percentage of gold precipitated on the cathode. Electrical precipitation is further considered later under the heading, "Electrical precipitation on copper plates."

W. Bettel gives the results of work by this process in the autumn of 1896 at two of the Transvaal mines, as follows:*— The "Bonanza" mill is designed to treat about 3,000 tons of tailings and concentrates per month. The pulp is roughly concentrated in spitzlutten, about 20 per cent. of sand (containing the pyrites, &c.) being separated and treated separately from the rest. There are 12 tanks, each holding 100 tons, arranged in two rows, one discharging into the other. Each charge of tailings is treated for 2½ days in an upper vat, and 5 days in a lower one. A typical charge of tailings was treated as follows:—8 tons of alkaline wash (with 0·008 per cent. KCy) were added at 9 a.m. on October 19, 1896. Between 12 noon and 2.30 p.m., 8 tons of weak solution (0·025 to 0·03 per cent. KCy) were run on. From 19th to 21st October, 34 tons strong solution (average 0·07 per cent. KCy) were used, and the charge was then drained and transferred to the lower vat, where, between 21st and 24th October, 72 tons of 0·07 per cent. KCy solution, 22 tons of 0·025 to 0·03 per cent., and 24 tons of 0·008 per cent. KCy ("alkaline solution") were run through, giving a total of 168 tons of solution to 100 tons of tailings.

A charge of 100 tons of concentrates was treated in the upper vat from 2nd to 7th October (5 days) with 13 tons of alkaline solution, 10 tons of weak solution, and 100 tons of strong

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[*] *Proc. Chem. and Met. Soc. of S. Africa*, Meeting held Nov. 21, 1896.

solution. After being drained and transferred to the lower vat, the charge was treated from 7th to 19th October (12 days) with 200 tons of strong solution, 15 tons of weak solution, and 30 tons of alkaline wash, or 368 tons of solution in all. The concentrates assay about 2 ozs., and the residues 2½ dwts., the extraction being about 93 per cent. In 39 charges of tailings, on the other hand, the average gold contents was 16⅓ dwts. per ton before treatment, and over 3 dwts. after treatment, including nearly ½ dwt. of dissolved gold, the extraction being 80·9 per cent.

The precipitation plant consists of four boxes, each 30 feet × 4½ feet and 3 feet deep, divided into two pairs in series connected with a dynamo giving 200 amperes with an E.M.F. of 8 volts. The resistance of the weak solution is considerable. Each box contains 23,960 square feet of cathode, so that the current used is 0·0041 ampere per square foot of cathode, and the rate of flow is 3 tons of strong solution per hour per box, and 1 ton of weak or alkaline solution. There are therefore 333 square feet of cathode surface per ton of strong solution, and 1,000 square feet per ton of weak solution passed through in 24 hours. The liquid after precipitation usually contains 1 or 2 dwts. of gold per ton, an average of 85 to 95 per cent. of the gold being precipitated. In cleaning up, the lead is melted in a reverberatory furnace, and sold to the Rand Central Ore Reduction Company.

In October, 1896, the total cost was 6s. 8·72d. per ton of ore, but was soon after reduced by the introduction of continuous haulage. The cost of cyanide was nearly 1s. 10d. per ton, the consumption being over 1½ lbs. per ton, the cost of lead was 1·2d. per ton, and the royalty was 1s. 6d. per ton.

At the May Consolidated Works, there were in 1896 17 vats of an average capacity of 200 tons each, and the plant is capable of treating 12,000 tons per month. The solutions used range in strength from 0·1 per cent. KCy for strong solutions, and 0·025 per cent. KCy for weak solutions, to 0·01 or 0·02 per cent. KCy for solutions for preliminary treatment. The concentrates contain from 8 to 12 dwts. per ton, and are treated for ten days; the tailings contain from 3½ to 4½ dwts. before treatment, and are treated for four days. The residues only contain from 15 grains to 1 dwt. of gold per ton. The working costs in the three months, August to October, 1896, were an average of 2s. 7·7d. per ton in treating 9,000 tons per month, exclusive of about 4d. per ton for interest and sinking fund for the cost of the vats.

The Siemens-Halske process made considerable progress on the Witwatersrand goldfield before the introduction of the Betty-Carter lead-zinc process, when its chief advantage over other methods lay in the possibility of using more dilute solutions for dissolving the gold, with a consequent reduction in the loss of cyanide, of about ⅓ lb. per ton of tailings (Bettel).

The Siemens-Halske process requires well-filtered clear solutions, which have been made alkaline, for the best work. The lead foil may be amalgamated by the use of mercuric chloride (Durant's method) to ensure the adhesion of the gold. Continuous precipitation was carried out at the Ferreira Slimes Works in 1898, some of the lead strips being taken out and replaced by fresh ones each day. There were four boxes in use, each 36 feet long by 12 feet wide, holding 1 ton of lead each, 1 lb. of lead being consumed per ton of slimes. The extraction was 80 per cent. of the gold. The solution flowed through the boxes at the rate of 15 to 20 tons per hour, the value on entering the boxes being 29 grains per ton, and on leaving 4 grains per ton.

After the introduction of the lead-zinc couple, the Siemens-Halske process was partly displaced by it, and in the beginning of 1899, out of 16 slimes plants on the Rand, only 4 were using the Siemens-Halske and 12 the Betty-Carter process.

Electrical Precipitation on Mercury or Amalgamated Plates.—In the *Molloy process*, precipitation was made on "sodium or potassium amalgam formed electrolytically from a solution of carbonate in contact with a bath of mercury." In this connection it may be noted that Messrs. MacArthur and Forrest experimented on the action of sodium amalgam, using a small tower filled with pieces of sodium amalgam, through which the solution of gold in potassium cyanide was allowed to trickle slowly. They came to the conclusion that sodium was not more efficacious than zinc, but was much more expensive. The saving of expense in the production of the bullion from the precipitate was outweighed by the cost of the sodium required to form the precipitate.

Electro-deposition of gold is used in the Siemens-Halske process (see p. 316) and the conditions necessary for success have been worked out and stated by the inventors of that method. One of the most important provisions is that the layers of liquid between the anodes and cathodes must be very thin in order that the ions may not be compelled to travel far before they reach the poles. Hence, when great quantities of liquid have to be treated, it is necessary for the electrodes to have a very large surface. Moreover, they should stand in a vertical position so that they may not become coated with slime settling from the liquids, if these are muddy. It follows that mercury is not suitable for cathodes.

According to experiments made by the author,* electrical precipitation on mercury, or on amalgamated copper plates, is only effective when the current density does not exceed 0·01 ampere per square foot of cathode. With 0·01 ampere or less per square foot of cathode, gold is precipitated and amalgamated simultaneously, the plates keeping in good condition during 24 hours use, while ½ oz. of gold per square yard of plate was deposited on them. With currents of greater density, part of the gold is deposited as a non-adherent black powder easily rubbed off, but not easily detachable by a jet of water, and not easily amalgamated. As the density of the current is increased, the percentage of non-amalgamable gold thrown down rises, until at 0·30 ampere per square foot about half the precipitated gold is not amalgamated.

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* T. K. Rose, *Trans. Inst. of M. and M.*, 1900, p. 369.

The dimensions of the cathodes must therefore be far greater than if they consist of lead, when currents of 0·05 ampere per square foot are used with corresponding increase in the speed of precipitation. Apart from the cost of the plates, however, the cyanide solution suffers loss by dissolving copper from the cathode. Nevertheless, copper plates have been used in several processes.

Among other suggestions for improving electrical precipitation may be mentioned that by Andreoli of using anodes made of lead peroxide, which do not waste or damage the solution, and cathodes made of iron; that by Croasdale of using amalgamated zinc plates for anodes;* and that by Cowper Coles of using aluminium for cathodes.†

The Pelatan-Clerici Process.—In this process, the finely crushed ore is agitated in vats (holding 25 tons each) with a solution of about 0·1 per cent. KCy. The bottom of the vat is lined with amalgamated copper plates covered with a layer of mercury. A current of electricity is passed from the sheet-iron agitators, forming the anodes, to the mercury bath, which is the cathode, so that gold dissolved by the cyanide begins to be precipitated at once. It was worked at the De Lamar Mill in Idaho in 1897,‡ using very thick pulp. The practice there seemed to resemble pan-amalgamation with the additions of an electric current and some cyanide in the water, but subsequently plain cyaniding was resorted to.§

The *Riecken process*, formerly used at the South Kalgoorlie Mill, was similar.

Chemical Precipitation on Mercury—*Gilmour-Young Process.*‖—This method has been employed at the Santa Francisca Mine, Nicaragua. The ore contains a very large percentage of clay, the slimes amounting to 70 per cent. of the ore. The ore is crushed by a No. 5 Krupp ball mill through a 30-mesh sieve, and is charged into Boss pans, 4 feet in diameter, in 2-ton lots, with 100 gallons of water, so as to form a thick pulp. From two to six bottles of mercury are added, and circulation continued until the globules can be seen dispersed through the pulp, and then the required amount of cyanide. After running for two hours, 10 lbs. weight of mixed copper and zinc amalgam are added, and the pan run for four hours more. The gold is dissolved in the cyanide and precipitated by the zinc and copper, only 10 grains of gold per ton remaining in solution.

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* *Eng. and Mng. Journ.*, Dec. 12, 1896.
† *Trans. Inst. of M. and M.*, vol. vi. (1898), p. 219.
‡ *Eng. and Mng. Journ.*, Aug. 7, 1897, p. 155.
§ *The Cyanide Process*, Bosqui, p. 136.
‖ *Trans. Inst. Mng. and Met.*, vol. vii. (1898), p. 63.

The pulp and solution are then discharged into a settler and diluted, and the mercury recovered in the ordinary way. The amalgam is used over again, and eventually a bullion 800 fine is obtained by retorting. In the twelve months ending June, 1898, 4,129 tons of ore were treated, the cost of chemicals per ton being as follows:—

	lbs.	s.	d.
Cyanide, ...	1·67	2	1·05
Caustic, ...	·65	0	·65
Copper sulphate, . . .	1·16	0	2·03
Cast-iron turnings, . . .	·25	0	·12
Zinc, ...	·39	0	1·17
Mercury, ...	·40	0	7·81
		3	0·83

The power required for treating 50 tons per day is 24 H.P., and the labour costs are low. The copper sulphate and iron turnings are used to make the copper amalgam. The zinc amalgam is made by pouring mercury into molten zinc. The mean extraction was 83·6 per cent. of gold and 63·9 per cent. of silver from ore containing 1 oz. 5½ dwts. gold and 2 ozs. 7 dwts. silver, the tailings containing between 4 and 5 dwts. of gold.

Precipitation by the Lead-Zinc Couple.—The chemistry of the precipitation of gold by zinc is discussed in the next chapter. It is shown there that pure zinc is less efficient by itself in precipitating gold than when in contact with certain other metals. The lead-zinc couple, which had been patented by MacArthur in 1894, was introduced at the Lisbon Berlyn Mine in Lydenburg by him,* and independently by Betty and Carter at the Crown Deep Slimes Plant in 1898.† The method adopted there to prepare the couple was to turn the zinc on a lathe as usual, wash the shavings, and immerse them in a solution containing 10 per cent. of acetate of lead, stirring them until they had acquired a dark-coloured hue, due to a coating of precipitated lead. The shavings were then transfered to the extractor boxes and at once covered with auriferous solution to prevent surface oxidation. Half of a Siemens-Halske box was used, the required space being 40 feet long, 6 feet wide, and

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* *Journ. Chem. and Met. Soc. of S.A.*, Oct. 1898, p. 144.

† *Ibid.*, p. 140.

5 feet deep, divided into nine compartments. The solutions were clarified by filtering through sand beds in two tanks, each 36 feet in diameter and 12 feet deep, and then pumped to a small wooden tank at the head of the zinc box, and thence through a 4-inch pipe to the extractor. The solution coming from the slimes contained 0·007 per cent. KCy, and was raised to 0·025 per cent. for the first four hours by the addition of KCy at the point where it entered the box. For the next six hours the solution was raised from 0·007 to 0·014 per cent., and after that a drip of cyanide was added at the head of the box, enough to raise the solution from 0·007 to 0·008 per cent.

Most of the precipitation took place in the first two compartments, but by degrees the zinc became less active, and after a few days became "dead," rose to the surface, and floated away. Fresh zinc was cut about every eleven days, and the spent zinc used in the ordinary weak solution boxes in the sands plant. About 14 or 15 tons of solution were treated per hour, the average time of contact being thus over two hours. The solution was reduced in value by passing through the boxes from 36 grains per ton to between 2 and 4 grains per ton. At other works, however, from 5 to 7 grains of gold per ton were found in the sumps.[*]

The gold slimes contain much lead, one sample, for example, containing 23 per cent. They were treated in various ways, but the best results were obtained by dissolving in sulphuric acid and melting the residues in clay-lined pots (to prevent reduction of the lead by the graphite) with a flux composed as follows:—

Borax, .	. 60 per cent.	Sand, .	. 11·5 per cent.
Nitre, .	. 19 ,,	Sodium carbonate,	7 ,,

With this flux, bullion was obtained 876 fine. In three months' work in 1898 an average of 94·6 per cent. of the dissolved gold was precipitated.

The raising of the strength of solution at the head of the box, especially at the beginning of the run, is thought by Carter to be essential, and is supposed to act beneficially by keeping the zinc clean.

In a run of two months in the beginning of 1899, the combined results from different mills were as follows:—59,932 tons of slimes of an assay value of 2·94 dwts. were treated, 70·15 per cent. actual extraction made at a cost of 3s. 3d. per ton.

One of the difficulties in the precipitation by zinc is the appearance of a white incrustation partly consisting of zinc ferrocyanide, which covers the zinc shavings and prevents further action. This does not appear when much free cyanide and alkali is present, but is sometimes found to be troublesome in the Betty-Carter process, and is doubtless the cause of the zinc becoming "dead." The danger of the formation of zinc ferrocyanide is, of course, increased with the increase of soluble salts of iron in the solution, and these should be prevented from forming as far as possible.

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[*] *Report of the State Engineer of the S. A. R. for 1898.*

Professor Prister and Dr. Bay[*] give the following analysis of the "white precipitate" from the Ferreira Deep Mill.

Ferrocyanide of zinc and potassium, $K_2Zn_3(FeCy_6)_2$,	10·45
Zinc cyanide, $ZnCy_2$,	22·73
,, hydrate, $Zn(OH)_2$,	54·79
Copper oxide,	0·40
Fe_2O_3,	1·00
Silica,	1·03
	90·40
Ca, SO_3, Al, Co, Mg, Au,	Traces.

A case in which a free milling oxidised cupriferous ore was treated with lead-zinc precipitation occurred at the Ulundi Mine, near Barberton. Here, according to A. H. Hartley,[†] the tailings from the stamp battery contained from 5 to 8 dwts., and the residues from the cyanide plant from 16 to 36 grains. The rate of flow through the zinc-lead boxes was 1 to 1½ tons per cubic foot of zinc in twenty-four hours, giving from ½ to ¾ hour contact. The addition of cyanide at the head of the boxes was found to be quite unnecessary. The usual strength of KCy was 0·009 per cent., but the solutions treated sometimes contained only 0·002 per cent., the precipitation remaining good. When the zinc became "dead," large additions of caustic soda or cyanide were necessary if it were desired to restore its efficacy. Raising the strength of the solution by 0·005 per cent. KCy was useless.

Generally speaking, however, with cupriferous ores there are advantages in using electrical precipitation.

Other Precipitation Processes.—Charcoal is used[‡] in a number of small plants in Victoria, but scarcely at all in other parts of the world. The charcoal is placed in small tubs and the solution passed in succession through a number of these. The precipitation of gold is said to bear no relation to the strength of cyanide, but no mention has been made of the precipitation of gold from solutions containing less than 0·04 per cent. of KCy. The sump liquors may contain as little as 3½ grains of gold per ton. The clean-up methods are similar to those employed in connection with the chlorination process. The cost is supposed to be about double that of precipitation by zinc.

The *Wilde process*[§] is named after its inventor, Prof. P. de

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[*] *Journ. Chem. Met. and Mng. Soc. of S. A.*, Sept. 1904, p. 77.

[†] *Chem. and Met. Soc. of S. A.*, May, 1899, p. 101.

[‡] J. I. Lowles, *Trans. Inst. Min. and Met.*, vol. iii. (1899), p. 190.

[§] *Revue Universelle des Mines, de la Metallurgie*, &c., Oct. 1, 1895. Brussels.

Wilde, who proposed to use a solution of 0·05 per cent. KCy to dissolve gold from its ores, then to acidify the solution with sulphur dioxide and to precipitate aurous and cuprous cyanides with sulphate of copper, leaving the solution to settle for twelve hours. This method involves the destruction of the cyanide, and so, in cases where it is profitable to do so, the solution is acidified and the cyanide is precipitated as $FeCy_2$ by means of ferrous sulphate before the gold is thrown down. Prof. Christy proposes to modify this process by acidifying the solution with sulphuric acid, volatilising the free hydrocyanic acid by passing air or steam through the solution and condensing it in potash or soda solution, which is allowed to trickle down through scrubbing towers. In this way 80 or 90 per cent. of the potassium cyanide is regenerated and no gold is precipitated. Freshly precipitated sulphide of copper or any cuprous salt is then stirred in when the whole of the aurous cyanide is precipitated in a few hours, and the gold may be recovered by treating the precipitate with a solution of sulphide of sodium and depositing the gold by electrolysis. Prof. Wilde subsequently proposed* to carefully neutralise auriferous cyanide solutions with dilute sulphuric acid, and then to add a solution of cuprous chloride in common salt, assisting the settlement of the precipitate by adding potassium chlorate or blowing air through the solution. The precipitate is then dried and calcined, and the oxide of copper removed by dissolving it in sulphuric acid, leaving the gold nearly pure. One objection would be that the ferrocyanides would all be precipitated as the copper salt, and the whole precipitate would, in South Africa, contain little more than 1 per cent. of gold. Although these processes are of theoretical interest, it does not appear likely that they will ever be of any practical importance.

Direct Treatment of Rand Ore.—Several attempts have been made to treat auriferous banket by the cyanide process alone, without first amalgamating it. In general, however, these have been unsuccessful. The coarser particles of gold, usually removed on the copper plates, require far too much time for their complete dissolution to admit of a high percentage of extraction being attained. The pyritic ores have not yet been proved to be amenable to this method, but with certain oxidised ores from near the surface of the ground, all the gold is in a very finely-divided state, and 75 to 80 per cent. is extracted by cyanide in two or three days. At the George and May Mining Company's works, the oxidised banket is very friable and easily disintegrated, and this was crushed dry, and treated direct with complete success in 1895. Here nearly half the ore was converted into intractable slimes if it was crushed in the battery, and in the last few months of 1894 only 55 per cent. of the gold had been extracted by amalgamation, followed by treatment with cyanide. In the method afterwards adopted the ore was crushed dry by a Gates crusher, and dumped at once into the vats, where it was treated as follows (Hatch & Chalmers):—The ore is saturated with a solution containing 0·06 per cent. of KCy, and this is displaced by a "strong solution" containing 0·28 per cent.; a "weak" solution containing 0·10 per cent. of cyanide follows, and is washed out with one containing 0·05 per cent. "About sixty hours are required for the whole treatment. Some lime is mixed with the ore, and on an average 0·89 lb. of cyanide of potassium is used per ton of ore treated." This ore is so friable that it is reduced to barren pebbles and loose sand by merely passing through the rock breaker. About 75 per cent. of the gold is extracted, the ore containing about 5¼ dwts. per ton. The total cost of mining and treatment is about 12s. per ton.

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* *Journ. Chem. and Met. Soc. of S.A.*, Feb. 1899, p. 43.

Treatment of Dry Crushed Ore in New Zealand.*—This method has been superseded by wet crushing, but was largely used prior to 1898. At the Waihi Mine, the ore was dried in brick-lined kilns, which were charged with alternate layers of wood and ore, crushed in Gates crushers and stamps, and conveyed to the leaching-vats.

At the old mill there were 32 wooden vats, 22·5 feet in diameter and 4 feet deep, with the necessary sumps, tanks, extractors, &c. At the new mill there are 10 rectangular concrete leaching-vats, 50 feet by 40 feet in area, and 4 feet deep.

The wooden vats hold a charge of 30 tons, and the concrete vats of 150 tons each. The depth of the charge in each case is about 24 inches. With a greater depth of material the difficulties attending percolation became too great for economical working.

After the charge is levelled, the strong solution, containing from 0·33 to 0·4 per cent. of cyanide, is run under the filter-cloth until it shows on the surface of the pulp. The balance of the solution is then run on the charge from above. The portion of solution remaining below the filter is also pumped up and delivered on the surface of the charge.

The strong solution (in the proportion of 1 to 3 of ore) is allowed to percolate for thirty to forty hours. The weak solutions (1 to 3 of ore, and containing 0·1 to 0·03 per cent. of KCy) and the water-washes are then applied in rapid succession, their filtration being assisted by an artificial vacuum of 20 to 25 inches of mercury. Each ton of ore is washed with 1 ton of water applied in four separate portions.

About five days are required for the washings, and the total treatment of each vat occupies about 6·5 days.

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* See paper by James Park, *Trans. Am. Inst. of Min. Eng.*, *Colorado Meeting*, September, 1899.

The residues are sluiced out through side-doors, one man emptying 150 tons in eight hours.

The solutions from the leaching-vats are clarified by running them through tow filters, whence they pass to the zinc-extractor boxes, which are of such a capacity that 1 cubic foot of zinc is provided for every ton of solution per day.

The consumption of cyanide varies from 2 to 2·5 lbs. per ton of ore treated, and that of zinc is about 6·5 ozs., costing 5d. per lb.

The cost of treatment per ton of ore was as follows:*—

	s.	d.
Drying the ore (labour and fuel),	1	0·60
Crushing, including maintenance,	1	3·34
Stamping, ,, ,,	1	10·69
Total milling costs,	4	2·63

CYANIDING COSTS AT NEW MILL.

	s.	d.
Labour,	0	4·46
Cyanide,	2	0·00
Zinc,	0	2·00
Filter cloths,	0	0·70
Clean-up and smelting of slimes,	0	4·10
Total,	2	11·26

The actual recovery is said to be 89 to 90 per cent. of the gold and 55 to 60 per cent. of the silver. The output for one month in 1899 was 7,715 tons, yielding 21,974 ozs. of bullion, valued at £24,244.

Use of Cyanide in the Stamp Battery.—Efforts have been made in South Africa and the United States to treat the ore direct from the battery instead of first passing it over amalgamated plates. Ore from the May Consolidated Mine, Johannesburg, was crushed by the African Gold Recovery Company in the latter part of the year 1892,† with cyanide solution instead of water, and led at once into the filtering tanks. The results are stated to have shown that the coarse gold resisted the attack of the cyanide for so long a time as to render the process uneconomical. A modification of this system has more recently been successfully applied in New Zealand and in South Dakota (see pp. 329-331).

At the Stewart Mine, Bingham, Utah, a combination of the amalgamation and cyanide processes is said to have been applied successfully to ore containing a mixture of coarse free gold and fine rebellious particles.‡ The ore is crushed and amalgamated in Huntington and Crawford mills successively, the "water" used being a solution of cyanide of potassium. After leaving the mills the solution and pulp are run into settling tanks and the liquid drawn off. It would appear likely that the decomposition of the cyanide in the mills would be excessive, but no details of the working have been published.

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* *Eng. and Mng. Journ.*, Oct. 8, 1892, p. 342.

† *Ibid.* ‡ *Ibid.*, April 15, 1893, p. 339.

At the Luipards Vlei Estate Mine* a somewhat different method was adopted, and pyritic ores were treated with success. The oxidised ores were crushed through ⅛-inch mesh screens in Gates crushers, and rolls, and the pyritic ores through screens with 300 holes per square inch. Both kinds of ore were treated in steel vats, 8 feet deep, the depth of ore being 7 feet. There were two rows of these vats, and double treatment was adopted.

Lime was added to the ore at the rock-breaker floor, in proportions varying from 1 to 2½ lbs. per ton. By this means it was thoroughly mixed in the different machines, and any lumps were broken up.

The actual cyanide treatment differed a little from the ordinary procedure, and followed lines first suggested by W. R. Feldtmann.

As soon as a tank in the first row was filled, a strong solution, 0·25 per cent. KCy, followed by two others, the last being 0·15 per cent., was pumped on to it. The time given to this treatment would be 66 to 70 hours. The solutions were not allowed to stand, but were drained off as soon as the tank was filled. The object of this was to allow fresh air to obtain access to the mass of damp sand. Each solution would be about 27 tons to the tank of ore (165 to 170 tons).

The ore was then transferred to the second row of tanks, having lost about 67 per cent. of its original assay value in this short time. It is probable that the abundance of air entangled in the dry sand, as compared with what would be held in a tank of sand settled with water, materially assisted the solution of the gold. Also the finer grains of the free gold would be quickly taken up.

A solution of 20 to 25 tons, not exceeding 0·20 per cent. KCy, was then pumped on to the transferred sands and drained off, the mass being allowed to remain damp for about 96 hours, when weaker solutions (0·15 to 0·10 per cent.) were used in continuous washes, making up a total of 75 tons per tank (2nd treatment). A water-wash of 20 or 30 tons completed this part of the process, which would last some 275 hours.

A careful series of moisture tests and measurements of solution sumps during the treatment of four tanks (680 tons) showed that the total loss of liquid in the treatment was 96 tons, or 24 tons per tank. The moisture in the discharged residues averaged 12·3 per cent., or, say, 20 tons per tank, the remaining 4 tons would be represented by evaporation from surfaces of sumps and tanks and by leakages. As the ore contained about 3 tons of water per tank, in the form of moisture when delivered from the mill storage bin, the actual consumption of fresh liquid equals 21 tons per tank, or $\frac{1}{8}$ ton (25 gallons) per ton of ore.

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* Franklin White in *Trans. Inst. M. and M.*, vol. vii. (1899), p. 124.

The solutions running from the first row of tanks carried from 13 to 32 dwts. per ton; those from the second row 3 to 4 dwts.; and the final wash 0·08 to 2½ dwts.

In all, 25,133 tons of ore were treated down to October, 1898, the average assay value being 8·28 dwts., and that of the residues 1·4 dwts. About 83 per cent. of the gold was extracted, the costs being as follows:—

Cost of Milling Pyritic Ore.

		s.	d.
White wages,	per ton,	0	10·995
Native ,,	,,	0	3·924
Stores,	,,	0	8·863
Lighting,	,,	0	0·485
Steam power,	,,	0	7·856
Electric power,	,,	0	0·408
Maintenance, &c.,	,,	0	7·322
Total,		3	3·853
Rock breaker cost,		0	9·747

Cost of Cyaniding.

			s.	d.
White wages,		per ton,	0	11·18
Native ,,		,,	0	7·20
Maintenance,		,,	0	3·45
Assay charges,		,,	0	3·60
Cyanide,	0·40 lb.	,,	0	7·70
Zinc,	0·31 lb.	,,	0	1·08
Lime,	2·55 lbs.	,,	0	1·15
Light and power,		,,	0	2·30
Stables,		,,	0	0·85
Coal, coke, and sundries,		,,	0	1·00
Total,			3	3·51

Summary of Cyanide Cost.

		s.	d.
Tanks and extractor house works,	per ton,	2	3·30
Clean-up and smelting, &c.,	,,	0	5·15
Discharging residues,	,,	0	8·36
Sundries,	,,	0	1·05
Total,		3	5·86

*Crushing with Cyanide Solution in New Zealand.**—At the Crown Mines Company's mill, a chalcedonic ore with practically no "mineral" or slimes, a small proportion of free amalgamable gold, and a large proportion of fine cyaniding gold is treated by

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* See paper by James Park, *Trans. Am. Inst. Mng. Eng., Colorado Meeting*, Sept., 1899.

(*a*) Crushing with cyanide solution,
(*b*) Direct cyanide treatment of mixed sands and slimes by percolation,
(*c*) Plate amalgamation of free gold.

The ore is hard and splintery, completely free from all impurities, and unlike most of the ores from the neighbouring mines, contains little or no silver, except what is alloyed with the gold. It is crushed in the Company's 60-stamp mill with cyanide solution in the mortars, about 2·5 tons of solution being used to 1 ton of ore. A 25-mesh screen is used, and the duty per stamp is about 2 tons per day. The slimes formed in crushing are said to amount to less than 5 per cent. The monthly output is about 2,750 tons. In June, 1899, 2,900 tons were crushed, yielding 3,037 ozs. 5 dwts. of bullion, valued at £6,069 14s. 8d., equal to a fraction under £2 per ounce.

The cyanide plant consists of 28 leaching-vats, each 22·5 feet in diameter and 4 feet deep. Much reticence is maintained as to the exact treatment, but the main features are understood to be as follows :—

The whole of the pulp from the 60 stamps is conducted by a launder to one vat, and allowed to discharge into the centre until the vat is about half-full. The pulp is then diverted to another vat, which is allowed to fill in the same manner. The mixed sands and slimes in the first vat are allowed to settle for an hour or two, after which the fairly clean top-solution is siphoned off into a collecting-tank, whence it is pumped up to two elevated tanks, from which the solution for the stamps is supplied. The pulp is again diverted into the first vat until the charge is complete. After settlement, the clear top-solution is again drawn off. In this way three or four vats may be in course of filling at the same time.

The settlement of the slimes is effected without the aid of lime by allowing the solution to percolate from the bottom of the vat during the filling and periods of settlement. The downward currents promoted by the draining from below are said to promote the settlement of even the finest material within a reasonable time. This is a point to be noted by cyaniders troubled with slimy products.

The mixed sands and slimes are treated by percolation in the ordinary way. The depth of each charge is about 30 inches, and the weight 40 tons. The strong cyanide solution is allowed to percolate from 30 to 40 hours, while the weak cyanide and water-washes are drawn off by the aid of an air pump.

The siphon used in the sand-vats consists of a length of 2·5-inch rubber hose, to one end of which is attached a short length of wooden batten, to keep it on the surface of the solution. The other end is fixed, inside the vat, to a short iron pipe passing through the side, about 18 inches above the filter-cloth.

In the extractor-room there are five precipitation-boxes of the ordinary pattern, divided into compartments by baffle-boards; and four zinc-towers, consisting of wooden boxes about 6 feet high and 30 inches square, set on end and connected in a series, like charcoal-towers. The solutions flow upwards through the zinc-turnings, the overflow being conducted in a pipe to the bottom of the next tower, and so on to the last.

The sands are sluiced out of the vats over a wide expanse of amalgamated copper plates, which catch a certain proportion of the free gold.

The actual recovery from all sources is said to vary from 84 to 87 per cent.; but the figures of cost are not obtainable.

*Crushing in Cyanide Solution in the Black Hills, South Dakota.**—Five mills are already at work using this system. The "red ores" are almost exclusively treated at present. The ore is crushed in stamp batteries in a weak cyanide solution containing from 1·3 to 2·2 lbs. KCy per ton (0·065 to 0·11 per cent. KCy), and having a protective alkalinity equivalent to from 1 to 1·5 lbs. of caustic soda per ton. The sands are then separated from the slimes by means of cone-classifiers, the sands cyanided by percolation, and the slimes by agitation and decantation. The battery screens range from 10 to 26 mesh, and the duty of the stamps is about 4 tons per head per diem. About 40 per cent. of the crushed ore consists of slimes, and some of the sand is left in the slimes. "Barren solution," free from gold, is used in cyaniding after the mill solution has been removed. The strength in cyanide of the two solutions is about the same. When the solutions are passed to the zinc boxes, fresh cyanide is added, bringing them up to about 0·15 or 0·20 per cent. in KCy.

At one mill the Moore process is used to shorten the time of decantation. In this process filter frames are lowered into the vat, and solution removed through them by vacuum leaching.

At the Dakota and Horse-shoe Mills zinc barrels instead of zinc boxes are used for precipitation.

The extraction at the mills varies from 65 to 75 per cent., about half of which is obtained in the battery. The cost of treatment is from $1.17 to $1.79 per ton, the total consumption of cyanide being about 1 to 1½ lbs. per ton.

Treatment of Concentrates.—Concentrates can be treated successfully by percolation, even if they consist chiefly of sulphides of iron, lead, zinc, &c. The time of treatment is in these cases often as much as three or four weeks, and the charges in the vats are sometimes drained and stirred up or transferred to other vats, in order to aërate them and expedite the dissolution of the gold. The use of Sulman's bromo-cyanogen, or other accelerator, is also often desirable or even necessary. This lengthy process is now being shortened by finely crushing the concentrates in tube mills before cyaniding.

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* See papers on "Crushing in Cyanide Solution in South Dakota," by C. H. Fulton, and "Cyanide Practice at the Maitland Properties," by John Gross. *Trans. Am. Inst. Mng. Eng.*, September, 1904.

Concentrates are also treated by agitation as, for example,[*] at the Woodstock Mill, New Zealand, where about 100 tons of concentrates per month are treated in this way. They have a value of £30 to £40 per ton, a large proportion of the value being in silver sulphide. They are treated by agitation with a 4 per cent. solution of cyanide for 36 hours. Two pounds of lime are added to every ton of concentrates. The charge weighs about 1·5 tons. Mr. Frank Rich, the superintendent, who adopted the present treatment, stated that the recovery varied from 90 to 94 per cent., at a cost of 18s. per ton for labour and material.

The Sulman-Teed Process.—Among other suggestions for rendering the presence of oxygen in cyanide solutions unnecessary, the most striking was that made by H. L. Sulman and Teed,[†] who use cyanogen bromide, $CNBr$. The addition of this substance to a solution of potassium cyanide makes it much more rapid in dissolving gold. They put forward the equation

$$CyBr + 3KCy + 2Au = 2K.AuCy_2 + KBr$$

in explanation of the action of their solvent, which is inoperative except in the presence of an alkaline cyanide. Cyanogen chloride and iodide give equally good results as far as rate of solution is concerned, but are not convenient for use on a large scale. In tests on concentrates, and on various complex ores, the results obtained by Sulman, Teed, and others were remarkably good, high percentages of extraction being obtained in a few hours from ores which yielded little or no gold to ordinary cyanide solutions in the same time.

Cyanogen bromide is efficacious in the treatment of tellurides without roasting. Certain tellurides of gold appear to be dissolved by it, although but little attacked by alkaline cyanides. The Diehl process (*q.v.*) depends on this for its success. The cyanogen bromide or bromo-cyanide process was also used at Deloro, Ontario, to treat auriferous mispickel ores.[‡] The consumption of cyanide was 0·9 lb. KCy and 0·45 lb. $CNBr$ per ton. The total cost of treatment was 6s. 3d. per ton, and 84 per cent. of the gold was extracted.

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[*] J. Park, *loc. cit.*

[†] *Proc. Institute of Mining and Metallurgy*, Feb. 1895.

[‡] See paper by H. K. Picard read before the *Institution of Mining Engineers*, May 20, 1898.

One of the pecularities of the gold bullion produced by the use of bromo-cyanogen is that it contains tellurium which is dissolved, presumably as a bromide, and precipitated by zinc. Gold bullion produced by ordinary cyanide does not contain tellurium, which is insoluble in cyanide. It is probable that cyanides of tellurium cannot be formed.

Cyanide Practice at Particular Mills.—*The Robinson Mill.*—At the Robinson Mine* there is a 60-stamp battery, pulverising 280 tons per day through a 40-mesh sieve, the ore containing nearly 2 ozs. of gold per ton. Of this amount 71·04 per cent. is caught on the amalgamated copper plates, 5·13 per cent. is extracted from the concentrates at the chlorination works, and 18·55 per cent. from the tailings at the cyanide works, while 5·28 per cent. is lost. The pulp from the battery is concentrated, and the tailings caught in pits and shovelled into cars and conveyed to the cyanide works, while the concentrates go to the chlorination works. There the concentrates are placed in heaps, assayed for sulphur, and mixed so as to make a uniform product for roasting. They are roasted in three furnaces, each 60 feet long, treating 600 to 800 tons per month. When roasted, the material is charged into ten circular chlorination vats with bottom discharge and luted rings and lids. The chlorination occupies four days, 6 lbs. of chlorine being used for each ton of concentrates. The gold is precipitated by a solution of ferrous sulphate, the mass being kept agitated by means of a jet of compressed air. The consumption of ferrous sulphate is from 10 to 15 lbs. per ton of concentrates. The gold is allowed to settle for some time in a series of large vats, and a complete clean-up takes place only four times a year.

At the cyanide works there are twelve vats each holding 75 tons. The treatment here also occupies four days, the output being 225 tons per day. The whole of the tailings are subjected to treatment, the coarse gold having been removed on the plates, and the pyrites by concentration, so that almost clean sand, containing from 8 to 10 dwts. of finely-divided gold, is left for the action of the cyanide. Of this from 1½ to 2 dwts. are left in the tailings, the extraction being over 70 per cent.

New Primrose Cyanide Works.†—At these works the so-called "double treatment" is used, the tailings being treated with cyanide in two vats in succession. The pulp from the battery is passed into hydraulic classifiers (*spitzlutten*), and coarse sands and pyrites, amounting to about 10 per cent. of the whole product of the stamps, are separated from the finer material. The coarser materials, which contain 15 dwts. of gold, are drained and treated for three or four days with a solution containing 0·3 per cent. of cyanide. They are then transferred to other vats and treated for a further period of three days with weaker solutions, which are finally displaced by water. The extraction amounts to 85 to 93 per cent., the consumption of cyanide being about 1½ lbs. per ton. The fine material, overflowing from the classifiers, is fed into a distributing tank and thence to other tanks in which the sand settles, while the slimes overflow and are carried away. The collected sands are then drained dry, the operation being expedited by the use of vacuum pumps, which afford a vacuum of 15 inches of mercury. A solution containing 0·04 per cent. of cyanide is then run on, and replaced by a stronger solution, which is allowed to stand for twelve hours, and then drained off and the sand removed to the ordinary leaching tanks. Here a solution containing 0·2 per cent. of cyanide is added, and allowed to stand for twelve hours; it is then replaced by weaker solutions, 0·08 and 0·04 per cent. being used successively, and finally water, rendered alkaline by milk of lime. An average of 78 per cent. of the gold is extracted, while, before the "double treatment" was introduced, only 55 per cent. was obtained. The capacity of the plant is 17,000 tons per month, and the working expenses average 4s. 1d. per ton. The consumption of chemicals per ton of ore is as follows:—Cyanide 0·98 lb., zinc 0·22 lb., and caustic soda 0·22 lb. The bullion recovered is worth £3 5s. per oz.

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* B. H. Brough, *Journ. of the Soc. of Arts*, vol. xli. (1893), p. 173. The description refers to the methods in use in 1893.

† The account given here is mainly based on the notes supplied by W. Bettel, consulting chemist of the New Primrose Mine, to Hatch & Chalmers' *Gold Mines of the Rand*.

The Plant at the Simmer and Jack Mine.—The old cyanide plant of the "Simmer and Jack," South Africa, is shown in Fig. 79.* Here it was necessary to raise the tailings, and this was done by hauling them up inclined platforms carried on trestles. A double gangway runs over the vats for filling. The leaching vats were the largest on the Rand goldfields, each holding 600 tons, being 40 feet in diameter with 14-foot staves, the staves, as well as the bottom, being of 9-inch by 3-inch material. There were eight discharge doors, 16 inches in diameter, in the bottom of each vat. The three solution vats were 26 feet in diameter, with 15-foot staves.

The Simmer and Jack plant in 1898, however,† consisted of thirty intermediate vats, 10 for concentrates and 20 for sand, each 24 feet in diameter and 11 feet 3 inches deep, with six bottom discharge doors; and thirty leaching vats, including ten for concentrates, each 30 feet in diameter and 10 feet deep. The Siemens-Halske method of precipitation was used, and there

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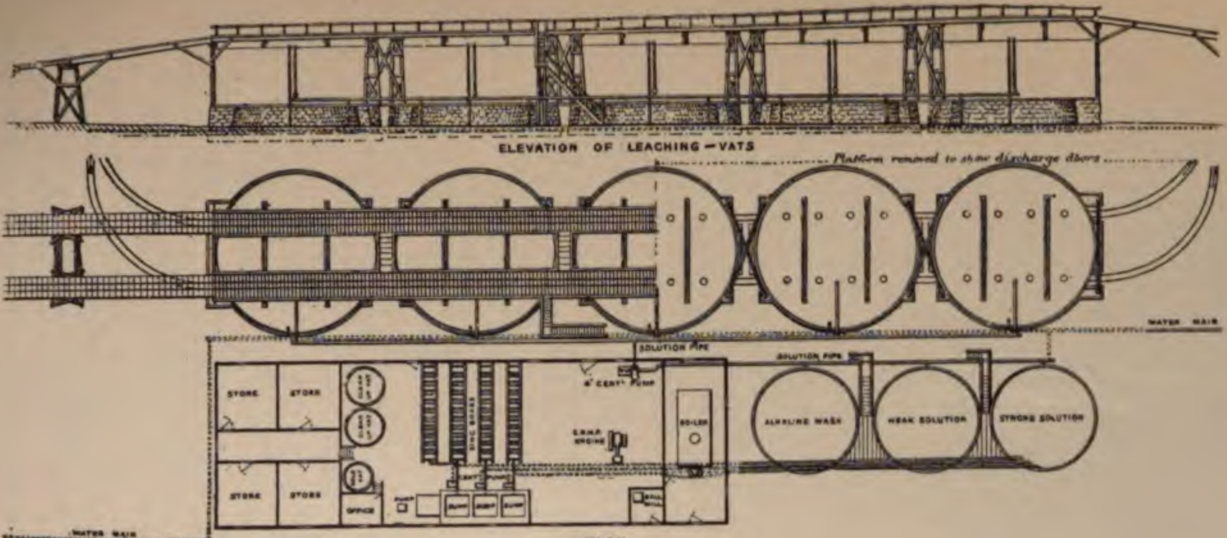
* Reproduced by permission from the paper by Butters and Smart in the *Proceedings of the Inst. of Civil Engineers*, vol. cxx., pt. ii., 1895.

† *Present-Day Metallurgical Engineering on the Rand*, by John Yates, 1898, p. 39.


Platform removed to show discharge doors
WATER MAIN
SOLUTION PIPE
SOLUTION PIPE
ALKALINE WASH
WEAK SOLUTION
STRONG SOLUTION
BOILER
E.H.P. ENGINE
4" CENT^L PUMP
ZINC BOXES
CENT^L PUMP
PUMP
PUMP
PUMP
PUMP
OFFICE
STORE
STORE
STORE
STORE
WATER MAIN

ELEVATION OF LEACHING—VATS

PLAN

Fig. 79.

were sixteen boxes, each 30 feet long, 4 feet 9 inches wide, and 3 feet deep. This plant took the product of 280 stamps, weighing 1,250 lbs. each.

*The Golden Gate Mill, Mercur, Utah.**—This mill is a steel structure, and is the largest cyanide works in the United States, having a capacity of about 1,000 tons per day.

The ore usually contains from $6.00 to $8.00 in gold per ton. The gangue consists mostly of limestone and shale containing from 40 to 60 per cent. silica and a little iron. In treating the ore it is divided into three classes—viz., oxidised, base, and mixed. The base ore contains from 1 to 6 per cent. arsenic sulphides (orpiment and realgar) and occasional "stringers" of cinnabar. The oxidised ore is crushed dry in rock breakers and rolls to ⅜ inch size, the mixed to ¼ inch, and the base to ⅛ inch, and the three classes are conveyed to separate bins.

The base ore is afterwards calcined in "Jackling" furnaces, which are of the straight line type, with hearths 120 feet long and 12 feet wide. There are two decks. The calcining is effected in the lower deck, and at the end opposite the feeding end there is provided an automatic elevator which raises the ore to the top deck. This is not covered, and is really the top of the furnace. The rakes, while cooling, gradually work the ore along towards the feeding end of the furnace, and the ore, cooled by radiation and water sprays which also keep down the dust, is removed from the furnace by a screw. The object of the Jackling furnace is to calcine the ore and then allow it to cool before delivering it to the bins. These furnaces calcine about 10 tons of ore for each ton of coal used, and have a capacity of about 70 tons per day each. The mixed ore is given a similar but shorter calcination, and each kind of calcined ore, and about an equal amount of raw oxidised ore, are carried by belt conveyors and discharged together into a charging bin.

The leaching department, 720 feet long and 65 feet wide, is fitted with 26 rectangular steel tanks, 25 feet by 50 feet and 5 feet deep, holding 260 tons each. The filter bottoms are of various types. When a tank is charged, about 1 foot of raw oxidised ore is placed in the bottom and levelled off, and the tank is then filled with the mixture of raw and calcined ores from the charging bin. If a tank is charged with calcined ore only, the ore becomes very compact when the solution is added, and hence less pervious to the solution.

The leaching is effected by percolation in from three to four days by means of a solution containing about 0·3 per cent. KCy, which is kept alkaline by adding 100 lbs. caustic soda to each 100 tons of solution before it is run on to the ore. The solution containing the gold from the leaching tanks runs into two "gold tanks," which are each 20 feet in diameter and 12 feet deep. From the gold tanks it is pumped into three "precipitating tanks," each of 30 tons capacity. Into each precipitating tank is inserted a ½-inch iron pipe, through which air under 20 lbs. pressure is forced, the pipe going nearly to the bottom of the tank. When the tank is about half filled with solution, the valve is opened, the air admitted, and zinc dust is thrown into the solution. As the tank fills more zinc dust is added. About 1⅓ lbs. of zinc dust is used for each ounce of gold precipitated. The air current keeps the solution in agitation, and the particles of zinc dust in suspension. The precipitation is rapid and very complete.

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* This account was written by Oliver Powers in the year 1900.

After the solution has been allowed to settle one hour it is drawn off to filters which collect the particles in suspension. Ten Stilwell-Bierce and Smith-Vaile filter presses are used, and from these the solution goes into the sump, a tank 24 feet in diameter and 12 feet deep.

It will be noticed that in the use of this process there is no possibility of mechanical loss due to precipitated gold going into the sump. The zinc dust used is a good grade of the bye-product produced in the distillation of zinc, containing from 85 to 90 per cent. of zinc. It costs 7 cents per lb., or rather less than sheet zinc.

Each precipitating tank and filter press is cleaned once each month. About one-half the values are found in the tanks, and one-half in the presses. The tanks, however, contain the greater portion of the impurities.

The slimes from the precipitating tanks and filter presses are taken to the refinery, where they are placed in shallow iron pans in muffle furnaces. The filter-cloths and paper from the presses are slowly burned, and as little zinc oxidised as possible to avoid mechanical loss in fumes. It is cooled and sampled, and then treated in a wooden dissolving tank, about 12 feet in diameter and 8 feet deep, with a dilute solution of sulphuric acid in the presence of nitric acid. Enough nitric acid is added to have an excess of red nitrous acid fumes. At the Golden Gate refinery, 1 lb. of nitric to 2 lbs. of sulphuric acid is found to be about the right proportion for the slimes.

In adding the sulphuric acid to the water sufficient heat is generated to start chemical action when the slimes are added. It has been suggested that nitre might be used instead of nitric acid, using an excess of sulphuric acid so as to convert the nitre into nitric acid.

The effect of nitric acid is to prevent the formation of hydrocyanic acid, and so avoid the danger of "boiling over" or of injury to the workmen. The cyanides are converted into cyanates, &c.

The solution and residue (gold and siliceous slimes) are then run through a filter press and washed thoroughly with hot water to dissolve out the cyanates and sulphates. The contents of the filter press are again placed in a shallow iron pan, and the filter-cloths and papers slowly burned out. The product is then cooled, the lumps broken to ½ inch or smaller, and melted in graphite crucibles in a small oil furnace with the following flux:—2 parts borax glass, 2 parts soda carbonate, and 1 part potash carbonate. The resulting bullion is about 950 fine.

The power was formerly supplied by a three-phase alternating current of 40,000 volts, transmitted 36 miles, but it was found that break-downs without warning occurred, and steam power is now used.

Cyanide Treatment of Cripple Creek Ores.—The following account of the method of treatment at the mill of the Metallic Extraction Company at Cyanide, near Florence, Colorado, was furnished in the year 1900 by W. F. Tindall, chemist of the works.*

The Cripple Creek ores consist of altered andesite, granite or phonolite, containing thinly disseminated iron pyrites and tellurides, mostly calaverite, associated with quartz and often with fluorite.† The gold is contained chiefly in the tellurides, and in the oxidised ore sometimes occurs as a visible ochre-coloured powder, "mustard gold." The pyritic ores contain rather less than 2 per cent. of sulphur.

The first attempt to treat Cripple Creek ores on a working scale by the cyanide process was in a small 10-ton mill, afterwards enlarged and called the Brodie Mill, situated close to the mines. This was a failure, owing to the difficulties of leaching the crushed ore. The mill was altered and enlarged in 1895, but even then the leaching was slow and unsatisfactory, and it was not until a dust separator had been devised that the oxidised ore (the only class treated at that time) was successfully dealt with. Using the experience gained in the Brodie Mill, P. Argall designed, erected, and started the Metallic Extraction Co.'s mill at Cyanide, near Florence, Colo., as a custom mill for the treatment of Cripple Creek ore on a large scale. Dust extractors were also used here until it was found that the leaching rate could be accelerated by roasting.

Several considerations made it desirable to place the mill at Florence, 38 miles from Cripple Creek, instead of in the district itself. The works were placed in direct communication with several of the mines, the freight charges being fixed at $3 per ton. Wages were high at Cripple Creek, where skilled labour cost $3 per day of eight hours, as against $2 at Florence, while in the latter district only $1.50 to $1.75 was paid for unskilled labour per day of ten hours. Fuel was easily obtainable at Florence, good steam coal costing $2 per ton at the works, slack coal 80 cents to $1 per ton, and residuum oil 50 cents per barrel. Oil was extensively used as fuel at Cyanide at first in preference to coal, for drying and roasting ore, but has now been discarded on account of the great demand for it for other purposes which has grown up of late years. About 3½ barrels are equal to one ton of good steam coal. The mill was placed on the bank of the river Arkansas, so that the water supply is unlimited. It is delivered at the mill by a ditch, and also by an electric pumping plant capable of lifting 400 gallons per minute from the river. The works were started in 1895, and subsequently enlarged and extended. The placing of the mill was justified by the subsequent removal of the El Paso Reduction Works from Cripple Creek to Florence after two years' work at its first location, and there are now seven cyanide or chlorination mills at Florence.

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* The account is supplementary to the account given by Philip Argall in the *Mineral Industry for 1897*, pp. 368-380.

† *Op. cit.*, p. 369.

Crushing Plant.—At the Metallic Extraction Co.'s Works the oldest mill, called No. 1, with a capacity of 130 tons per day, was formerly used exclusively for oxidised ores, these being crushed to 40-mesh size with ½ per cent. of lime, and treated direct with cyanide, without roasting. Mills Nos. 2 and 3, each with a capacity of 150 tons per day, are used for crushing the unoxidised ores which require roasting before treatment with cyanide.

The ore is crushed in rock-breakers to pass a 1-inch ring, and distributed over a long series of storage bins by an automatic belt distributor, thus ensuring a material of even value. Ores, containing as much as 3 or even 5 ozs. of gold per ton, are bought for cyanide treatment, but all ores are distributed in this way to form a mixture containing 1½ ozs., as this has been found to be the value best suited to the treatment.

The mixed ore is carried by pan and belt conveyors to the Argall multiple tube dryers, from thence to rolls where the oxidised ore is crushed to 40-mesh size and the sulphide ores to 30-mesh size, and then stored in bins for roasting.

The rolls used for fine crushing are very heavy high-speed rolls.

Throughout the crushing and fine grinding the product is screened after every operation. As far as possible all machinery is automatic; in fact the ore is but once handled, and that when it is being shovelled out of the railroad cars into the crushers. Each mill is fitted with one pair of 36-inch geared rolls for coarse crushing, and three pairs of 26-inch belted rolls for fine crushing, this plant of four pairs of rolls being sufficient for the reduction of 160 tons of ore per day to 30-mesh size.

Roasting Plant.—This consists of one Ropp straight line furnace of 100 tons capacity, and six Argall furnaces of 50 tons capacity each. The pulp is trammed from the storage bins and dumped into the hoppers of the roasters. This could be more cheaply done by conveyors. The furnaces discharge the roasted product into conveyors which carry it to the coolers, where it is automatically cooled; and from thence it is conveyed to the leaching tank. Connected with the mill dryers and roasters are two large fans discharging 10,000 cubic feet of air per minute, whereby all the fumes from the roasters, dryers, and roaster buildings are forced through an extensive system of flues, and when thoroughly cool through a Bartlett Bag House out into the open air. At first cotton bags were used, but owing to the quantity of steam present, as much as 20 tons of which are sometimes driven off in a day in the four dryers, the SO_2 very quickly rotted the cloth. Then flannel was used and found to be better, but the cotton thread used to sew the bags together gave out. Flannel bags sewn with worsted are the best for this purpose, and if the temperature is kept below 300° F., and moisture is kept low, the bags will last a very long time. The supports must be constantly watched to keep them from rusting out.

The initial cost of the bags is a great drawback, but this soon loses its significance when the saving in values is taken into account.

Coolers of two types are used. One is the water-tube revolving cooler, the reverse of a roaster, the ore passing between cold water tubes. The other kind is the water-jacketed trough conveyor. The ground lime (about 10 lbs. per ton) is added to the ore as it passes along the conveyor, and is thus intimately mixed with it, a condition which is essential to low consumption of cyanide.

The roasters are all fitted with automatic self-feeding stokers. The slack coal is brought and the ashes taken away by conveyors, so that nothing is done by hand; similar methods being now adopted in all the mills in the district.

A working drawing of a plant of three Argall furnaces and an Argall cooler, arranged for automatic work, is shown in Plate VII. It gives a general idea of the method. There are 13 Argall furnaces and 9 dryers now in use on Cripple Creek ore. The dryers are most successful and their capacity is

No. 1, ...	80 to 100 tons per day.
No. 2, ...	150 to 200 ,,

Treatment of Dust.—The dust from the flues is returned to the roasters, but the bag dust, which contains much tellurium and gold, was for a long time sold to the smelters at Pueblo. They bricked it and mixed it in the furnace charges, but their charges were so high that it became necessary to devise some method of treatment for it at the mill. A method involving bricking, roasting in kilns, coarsely crushing, and leaching with cyanide, was tried, but without success. The latest method, and one that gives remarkably good results at a comparatively low

PLATE VII

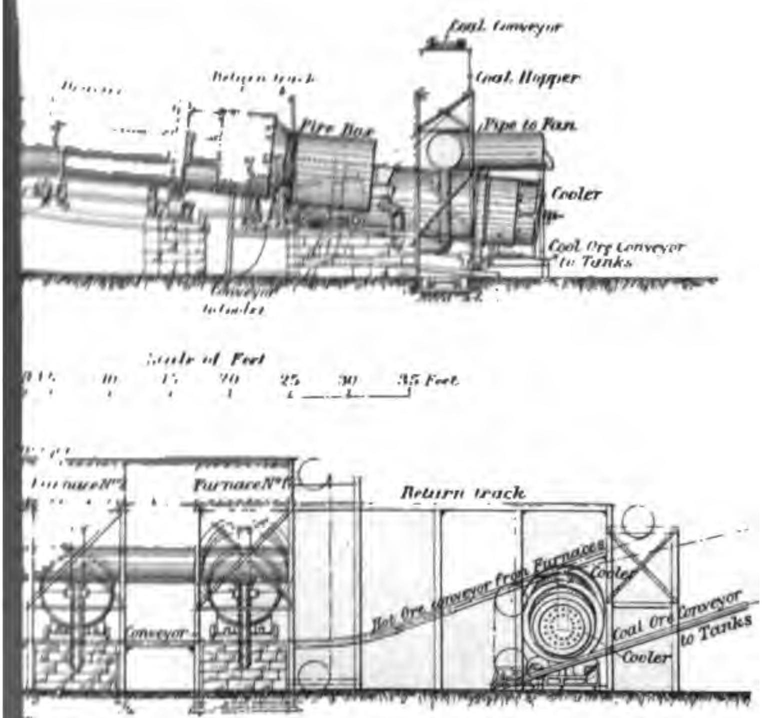
Coal Conveyor
Return track
Coal Hopper
Fire Box
Pipe to Fan
Cooler
Cool Ore Conveyor to Tanks
Conveyor to Cooler
Scale of Feet
10
15
20
25
30
35 Feet
Furnace No. 2
Furnace No. 1
Return track
Hot Ore conveyor from Furnaces
Cooler
Conveyor
Cool Ore Conveyor to Tanks
Cooler

Fig. 80.

he bag dust in a large muffle furnace in a still then to mix as much with the other roasted ore vith efficient and rapid leaching.

passes along the conveyor to the tank building, et very hot, it is sprinkled with strong cyanide 12 lbs. per ton of 2,000 lbs.) which reduces the nt by 20 per cent., and usually gives a better xtraction. The ore contains 4 or 5 per cent. of his has been found to be enough to give the with little, if any, increased cost caused from the in handling.

Plant consists of four large buildings containing

0 ton tanks, . .	Total capacity,	4,500	tons.
0 ,, ,, . .	,,	800	,,
0 ,, ,, . .	,,	600	,,
25 ,, ,, . .	,,	350	,,
20 ,, ,, . .	,,	40	,,
		6,290	,,

te tank capacity of about 6,300 tons is equal to y leaching capacity. The tanks are all made of y in depth and size, the first installed being only hile those in the second building were 50 feet in 5 feet deep, and those in the latest buildings neter and 6 feet deep. The increase in depth was e but little reduction in the rate of leaching, and initial cost per ton considerably. All the tanks drains, bottom discharge valves, and wooden false e wooden false bottoms are 5 or 6 inches from the and are made of inch lumber laid on radial strips, inches or 2 inches × 6 inches, placed edgeways. oms are perforated with $\frac{3}{4}$-inch holes, and covered re matting, covered with 10-oz. duck or canvas.

tank has strong solution run into it until the solu- inches above the canvas. Filling with ore is then nd solution is run in at the same time from below, out the time the tank is filled with ore the solution ing above the top. The ore is sampled as it is the tank, and duplicate assays for gold and silver The first solution is allowed to stand on the ore for hours and then drawn off, and more added and This is repeated with strong solution until the ore ssay to have lost about two-thirds of the gold con- which weak solution is used until the extraction is ow that nothing can be gained by longer treatment; nk charge is washed with several waters, drained, l sluiced out. The sampling is all done by a large 7 feet long, a number of holes being drilled at different places in the charge from top to bottom. Two samples are taken at each time, except in the case of "finals," and then four are taken.

A very careful record of each tank is kept, and no pains or work spared to ascertain the state and condition of each individual tank charge.

The accompanying scheme gives some idea of the leaching of a tank charge.

Tank charge, No. 1.
Time taken in filling, 50 hours.
Weight of ore, 434 tons.
Moisture, 5·4 per cent.
Weight of dry ore, 410 tons.

Date.	Tons of solution added.	Strength of Sol. added, per cent. cyanide of potassium.	Date.	Strength of Sol. withdrawn, per cent. cyanide of potassium.
Jan. 1	150	0·66	Jan. 3	0·73 to 0·68
,, 4	62	0·70	,, 4	0·63 ,, 0·63
,, 5	87	0·62	,, 5	0·62 ,, 0·62
			,, 6	0·70 ,, 0·65
	299	strong	,, 7	0·67 ,, 0·58
			,, 8	0·54 ,, 0·53
,, 6	20	0·35	,, 9	0·59 ,, 0·47
,, 7	30	0·35	,, 10	0·27 ,, 0·22
,, 8	52	0·37	,, 11	0·162 ,, 0·142
	102	weak		
,, 9	107	water		
,, 10	100	,,		
,, 11	63	,,		
	270	,,		

The assay of ore as charged into the tank was gold 0·90 oz., silver 0·50 oz. per ton, and the final tailings contained gold 0·06 oz., silver 0·20 oz. The percentage of precious metals extracted was gold 93·33 per cent. and silver 60·0 per cent.

The loss of cyanide was 1·75 lbs. per ton of ore treated. This charge took 303·15 hours from the time the men began to fill the vat until the tank was empty and ready for another charge.

The method of discharging the tailings is cheap and efficient. Each tank has some six, eight, or more sluice valves in the bottom, and when the ore is ready to be discharged these are opened.

A man, dressed in rubber boots and coat, then gets into the tank and sluices the tailings out of the tank by means of a large fire hose, throwing a stream of water at 120 lbs. pressure.

All the tank sluice valves are connected with a central station, by means of wide sluice boxes, which carry the sludge to a large centrifugal pump about 3 feet in diameter.

The pump elevates the sludge into a wide sluice box on a high trestle, along which it flows out of the works on to the dump. The sand-pump station is capable of handling 100 tons of sand, or 300 tons of sludge per hour, and has often exceeded this; but, as a rule, the sluicing is not hurried, for the tank capacity is so large as never to cause a shortage of tank room. There are two of these pumps always ready in case of a breakdown, and they can be driven either by an electric motor or a small steam engine.

The screen analysis of the tailings from the above charge was—

Coarser than	40-mesh	.	19·0 per cent.	0·07 oz. Au.
Between	40 to 100 ,,	.	40·25 ,,	0·06 ,,
,,	100 ,, 200 ,,	.	16·0 ,,	0·06 ,,
Finer than	200 ,,	.	24·75 ,,	0·07 ,,

This shows a very even leach, for there is very little more in the slimes than in the main bulk, and as only 19 per cent. was coarser than 40-mesh, it would not pay to have crushed this originally to 40-mesh instead of to 30-mesh size in order to have gained the additional 0·01 oz. of gold.

Precipitation of Gold.—There is one precipitation house and refinery common to all the leaching buildings. The method of precipitation is by zinc shavings, which, with slight modifications, answers admirably for all strengths of solution, the strong solutions ranging from ·70 per cent. KCy to ·25 per cent. KCy; the weak from ·25 per cent. KCy to ·10 per cent. KCy, and the wash-waters from ·10 per cent. KCy to ·02 per cent. KCy and less.

Each kind of solution is conducted to a separate receiving tank, to ensure that each zinc-box gets its proper share of solution, and then passes slowly through the zinc-boxes into the sumps and to the storage tanks after being strengthened to the required standard.

There are three precipitation boxes for each kind of solution. The "strong" boxes are very large, each being capable of dealing with 120 to 150 tons of solution per day, but the weak and the wash-water boxes can deal with only 50 tons each.

The zinc-boxes are made of sheet steel. Fig. 72 (p. 294) shows the details and dimensions of a 50-ton box used for weak solution. Those used for strong solution and wash-water are similar in principle, but differ slightly in detail and capacity.

The precipitation in the boxes generally reduces the amount of gold contained in a ton of solution as follows;—Strong, 1·25 oz. Au to 0·01 oz. Au; weak, 0·30 oz. Au to 0·02 oz. Au, and the wash-water 0·080 oz. Au to 0·002 oz. Au.

The zinc shavings are turned by hand on a lathe from discs of zinc of 15 gauge, 12 inches in diameter, with a 1-inch hole for the mandrel. They are supplied by the Illinois Zinc Works.

Great care is taken in this department in order to get as thin a shaving as possible with a minimum of dust, for on this in a great measure depends the purity of the resultant gold slimes. A clean-up is made of the strong boxes once a week, of the weak boxes once in two weeks, and the wash-water once a month.

Cleaning-up.—In order to make a "clean-up," the solution is turned off each box in sequence. The coarse zinc is then removed by hand, the workman wearing rubber gloves. Before taking the coarse zinc out, it is rinsed by "sousing" it up and down in the solution left in each compartment.

The solution and slimes left in the boxes are then drawn off by opening valves at the bottom of each compartment and conducted to a sheet steel vacuum filter tank, fitted with a false bottom, on which are laid several thicknesses of cotton flannel. There is also a suction pump connection underneath.

When the compartments of the zinc boxes have been washed with a jet of water until all slimes have been driven into the filter, the valves at the bottom of the compartments are closed, the old zinc is replaced, beginning at the top compartment and finishing with any new zinc that is wanted in the lower ones, and the box is ready for another week's run.

The slimes in the filter tanks are drained, washed, and sucked free of water by the suction pump, then placed in shallow iron pans (3 feet square), and roasted at a red heat in a muffle furnace over night.

It requires in this way about 0·9 lb. zinc to each ton of ore treated, or 0·5 lb. zinc to each ounce of bullion precipitated.

The roasted zinc slimes are cooled and treated in a lead-lined sheet-steel pan, with common commercial hydrochloric acid (one part acid to two parts water) until all action stops.

The resultant slimes of gold and silver are filtered, washed, and filtered again as dry as possible, and after being dried in shallow iron pans, are ready for smelting.

This smelting is done in large graphite crucibles, the mixture used having approximately the following composition :—

Acid treated slimes, ...	30 parts.
Soda carbonate, ...	14 ,,
Borax glass, ...	12 ,,
Fluorspar, ...	4 ,,
Sand, ...	8 ,,

The sand is increased if the slimes contain more zinc than usual, but, as a rule, the acid dissolves most of the zinc and in this way greatly reduces the bulk of the slimes. In one case thirteen pans of roasted slimes (one week's clean-up) were reduced to five pans of treated slime ready for drying by using about 750 lbs. of commercial hydrochloric acid.

The lumps of bullion are taken from each casting and all remelted and cast into ingots or "bricks" of about 1,000 ozs. each. From such a week's clean-up as described above two such ingots would be obtained. The ingots are sampled as accurately as possible and sold to one of the mints.

By this method of treatment, and by "working" or refining the molten bullion with suitable fluxes in the final melt, the grade is kept up and the gold is usually malleable. This bullion was considered for years to be the best grade of cyanide bullion sent to the Denver mint, and little difference is found either in gold or silver between the assays made at the mill and those made at the mint.

As a rule the bullion is about 850 fine in gold, 100 in silver, and the rest consists of zinc, copper, and tellurium.

The slags are partially cleaned by re-melting, the mixture used consisting approximately of—

Slag, . . .	100 parts.	Lead oxide, . .	8 parts.
Soda carbonate, .	20 ,,	Fluorspar, . .	1 ,,

The slags from this melting are saved and sold to the Pueblo smelters once or twice a year.

The lead bullion is saved and sold periodically to the refiners.

Testing the Solutions.—The strength of the cyanide solutions is tested by silver nitrate, using 10 c.c. of the strong and weak solutions respectively (diluted to about 100 c.c. with water) and taking 100 c.c. of wash-water undiluted. In each case a few drops of a 5 per cent. solution of potassium iodide are added. When the titration is done very rapidly and carried to a pronounced opalescence, this method in the hands of ordinary tankmen gives results fair enough for comparison in commercial mill work.

As a careful check is kept on all the ore and solution in every stage of the treatment, the number of gold and silver assays and other determinations on control samples, headings, tailings, "finals," flue dust, bag dust, slimes, gold bullion, lead bullion, slags, sweepings and solutions is necessarily very large.

Strong and weak solutions are assayed for gold by evaporating 1 A.T. in a small lead tray and then scorifying and cupelling the tray; but this method does not give satisfactory amounts to weigh in the case of wash-waters. If 20 or 30 A.T. are taken for evaporation, it must be begun in glass or porcelain vessels and finished in a lead tray after having been very carefully transferred. This entails much tedious work if some 15 or 20 determinations are required per day. Two rapid methods have been devised and used at Cyanide for this determination.

In one method a reagent of zinc dust coated with lead is prepared as follows:—500 grammes of zinc dust known as "blue powder" are mixed with water; to this is added an aqueous solution of 25 grammes of lead acetate, and the whole thoroughly stirred. When the zinc becomes coated with lead 20 A.T. of wash-water are placed in a bottle, and to this is added about 10 grammes (usually measured in a wooden mustard spoon) of the lead-zinc mud. The solution is agitated in the bottle for an hour or more, the lead-zinc mud allowed to settle, and all but 50 c.c. of the solution decanted through a filter leaving the lead-zinc in the bottle. Ten c.c. of oil of vitriol are added, and, when all action has ceased, the residue is thrown into the same filter and washed with water. When drained, the wet filter and residue are placed on a large scorifier with 15 grammes of sand, burnt in the front of the muffle furnace, and then assayed by the fusion method. This gives very good results if the "blue mud" is in excess, and if not too much zinc is left in the residue to be fluxed off with the sand.

In the second method 20 A.T. of "wash-water" are placed in a large beaker, and 10 to 15 grammes of the dry "blue powder" are shaken into it from the tip of a spatula, 50 or 75 c.c of strong commercial H_2SO_4 are added and the whole vigorously stirred and allowed to stand for one hour, or until all decided action has ceased. The contents of the beaker are then filtered, the residue is washed, and the wet filter is burnt on a scorifier with ½ A.T. sand and fused in a small crucible with litharge, soda carbonate, borax, and fluorspar, and the assay finished in the ordinary way. This method has been found to give very fair results with the "wash-water" solutions, and, besides not taking very long, allows of a large number of determinations being made at the same time.

These methods are technical methods, in which extreme rapidity is aimed at rather than extreme accuracy. However, they give fairly good comparative results.

All the methods in use at this mill have stood the test of time, and some of the modifications in the practice now employed are kept secret, a necessary consequence of the close competition of other custom mills in the district. The cost of treatment is not published, but $19 are paid per fine oz. of gold in the ore and nothing for silver; the charge for treatment is $7 per ton, including railroad charges for freight, which amount to $3 per ton.

Results of the Cyanide Process.—The success of the process in South Africa, where a new industry has been created, has been complete. The rapid progress which has been made there may be judged from the production of gold by cyanide in the years following its introduction in 1890:—

Year.	Total Output at Rand.	Production by Cyanide.	Percentage of Total Output.
	Ozs.	Ozs.	Per cent.
1890,	494,817	300	0·06
1891,	729,238	35,000	4·8
1892,	1,208,928	176,231	14·6
1893,	1,476,502	304,498	20·6
1894,	2,023,198	587,388	29·0
1895,	2,278,110	677,435	29·7
1896,	2,280,892	700,000	30·7
1904,	3,779,621	1,355,836	35·9

There were at first large accumulations of tailings to be dealt with, but since the year 1894 little but tailings or concentrates coming direct from the stamps have contributed to swell the output.

The importance of the process to the mining industry is not entirely represented by the output given above, as few of the mines could work at a profit if they depended merely on the gold extracted on the plates. The profit of, say, 13s. per ton derived from the tailings converts the mine from a losing into a paying concern in a large number of cases, and this profit could not be earned without the cyanide process.

In other parts of the world, the gold-mining industry is almost equally dependent on the cyanide process.

CHAPTER XVII.

CHEMISTRY OF THE CYANIDE PROCESS.

Action of Potassium Cyanide on Gold and other Metals.—Dr. Wright of Birmingham discovered in 1840 that metallic gold is soluble in potassium cyanide, and Elkington's patent, taken out in the same year, was partly based on this discovery. Bagration, in 1843,[*] studied the action of cyanides on plates of gold, and announced that they were slowly dissolved. Faraday, in 1857,[†] pointed out that gold-leaf is dissolved by a dilute solution of the salt, and also showed that if the gold floats on the surface of the liquid, so that one side of the leaf is in contact with the air, while the other is bathed by the solvent, the action is much more rapid than if the metal is completely submerged. Elsner had previously furnished some evidence[‡] that the presence of oxygen is required for the solution of the gold.

On evaporating the solution, colourless octahedral crystals of auro-potassium cyanide, $KAuCy_2$, are formed, which may be viewed as being a double cyanide, produced as follows:—

$$4Au + 8KCy + O_2 + 2H_2O = 4KAuCy_2 + 4KOH.$$

The equation for the solution of silver is similar.

Theoretically, therefore, 130 parts by weight of KCy in the presence of eight parts of oxygen suffice for the solution of 196·8 parts of gold. This has been proved by J. S. Maclaurin[§] to be the case in all carefully conducted experiments. The amount of oxygen dissolved in liquids not specially prepared, to say nothing of that contained in a porous mass of pulverised ore, is consequently enough for the solution of great quantities of gold.

According to G. Bodländer of Clausthal,[‖] the chemical action in the dissolution of the gold is as follows:—

$$2Au + 4KCy + 2H_2O + O_2 = 2KAuCy_2 + 2KOH + H_2O_2$$

and subsequently

$$4KCy + H_2O_2 + 2Au = 2KAuCy_2 + 2KOH$$

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[*] *Bull. de l'Acad. des Sciences de St. Petersbourg* (1843), vol. ii., p. 136.

[†] *Roy. Inst. Proc.*, vol. ii., p. 308.

[‡] *Erdm. Journ. Prak. Chem.*, vol. xxxvii. (1846), pp. 441-446.

[§] *Journ. Chem. Soc.* (1893), vol. lxiii., p. 724.

[‖] *Zeit. für angewandte Chem.*, 1896, p. 583.

Bettel suggests the following in place of this :—[*]

$$\begin{array}{ll}(1) & 6KCy + 2Au + O_2 + 2H_2O = KAuCy_2 + KAuCy_4 + 4KOH \\ (2) & KAuCy_4 + 2Au + 2KCy = 3KAuCy_2\end{array}$$

Evidence has been adduced that a substance is formed reacting like H_2O_2, but it does not follow that the actions represented by the equations given above are not limited to an insignificant part of the whole mass. MacArthur has cited experiments[†] to show that gold in ores can be dissolved by potassium cyanide in the absence of oxygen, and Bettel[‡] found this to be the case if the crushed ore contains basic ferric sulphate, by which potassium ferricyanide is formed, the reactions being expressed thus—

$$\begin{array}{ll}(1) & Fe_2(SO_4)_3 + 12KCy = 3K_2SO_4 + K_6Fe_2Cy_{12} \\ (2) & Fe_2(SO_4)_3 + 6KCy = Fe_2(HO)_6 + 2K_2SO_4 + 6HCy\end{array}$$

Bettel and Marais also found in 1894 that gold leaf would not dissolve in a solution of potassium cyanide from which the air had been expelled by the passage of a current of hydrogen, and that the addition, under these circumstance, of either potassium dichromate, chromate, chlorate, perchlorate, nitrate, or nitrite or of ferric hydrate or bleaching powder, did not enable the gold to dissolve. The addition of pyrolusite gave a doubtful result, and lead dioxide caused very slow dissolution of the gold. On the other hand, gold dissolved slowly if chlorine, iodine dissolved in potassium iodide, or ferric chloride were added; rapidly, if bromine were added; and decidedly, if potassium ferricyanide or permanganate, sodium dioxide, hydrogen peroxide or barium dioxide were added. It is clear, therefore, that certain oxidisers are ineffective, and in practice, when air is not used, potassium permanganate, sodium dioxide, and bromine (Sulman-Teed process), have been mainly employed to assist in the dissolution of the gold. In general, artificial oxidation is not resorted to unless there is some reducing agent present in the ore or the water which absorbs the oxygen. It was formerly supposed that any oxygen coming in contact with the solution would be at once taken up by the cyanide, forming cyanates, but it is now well known that oxygen and cyanide remain side by side without rapid union between them taking place.

It has long been remarked that a small percentage of a soluble sulphide present in the cyanide solution greatly delays the dissolution of gold. Doubtless this is partly owing to the abstraction of oxygen from the solution by the sulphide, for gold sulphide is freely soluble in KCy, so that the surface of the metal is kept free from sulphide. Bettel, however, points out* that silver sulphide is far less soluble than gold sulphide, and that if native gold containing 20 per cent. of silver is treated, a film almost insoluble in [dilute] cyanide solutions may be formed. It is certain that some specimens of gold leaf dissolve with great difficulty if they have been previously dipped in sulphide solutions, or if traces of soluble sulphides or sulphocyanides are present in the solution. The difficulty disappears if the sulphides are removed, either by being precipitated with lead salts, or by the action of certain oxidisers.

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[*] *South African Mining Journal*, May 8, 1897.

[†] *Journ. Soc. Chem. Ind.*, 1890, p. 7.

[‡] *S. African Mng. Journ.*, *loc. cit.*

The voltaic order of the metals in different solutions of cyanide of potassium is given in the following table :—†

1 part KCy in 8 parts water. ‡	1 part KCy in 160 parts water. §		1 part KCy in 3½ parts water. §	
	At 50° F.	At 100° F.	At 50° F.	At 100° F.
+ Zn-amalgam.	+ Al.	+ Mg.	+ Mg.	+ Mg.
Zn.	Mg.	Zn.	Zu.	Al.
Cu.	Zn.	Cd.	Cu.	Zn.
Cd.	Cu.	Al.	Al.	Cu.
Sn.	Cd.	Co.	Cd.	Cd.
Ag.	Sn.	Cu.	Au.	Sn.
Ni.	Co.	Ni.	Ag.	Au.
Sb.	Ni.	Sn.	Ni.	Ag.
Pb.	Ag.	Au.	Sn.	Ni.
Hg.	Au.	Ag.	Hg.	Hg.
Pd.	Hg.	Pb.	Pb.	Pb.
Bi.	Pb.	Hg.	Co.	Co.
Fe.	Fe.	Sb.	Sb.	Sb.
...	...	...	Te.	Te.
Pt.	Pt.	Bi.	Bi.	Bi.
Cast iron.	Sb.	Fe.	Fe.	Fe.
...	...	Te.	...	...
– Coke.	Bi.	Pt.	Pt.	Pt.
...	Te.	...	...	...
...	– C.	– C.	– C.	– C.

The position of the metals in the tables denotes their relative tendency to dissolve under the given conditions. When two metals are in contact, the negative metal assists the positive metal to dissolve, and is partly protected by it from the attack of the solvent. Conversely a metal tends to precipitate from solution another metal that is negative to it, and to replace it in solution. It may be deduced that mercury is inapplicable as a precipitant for gold from solution, and that aluminium, copper, or zinc could

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[*] *Loc. cit.*

[†] See also paper on the "Electromotive Force of Metals in Cyanide Solutions," by S. B. Christy. *Trans. Am. Inst. Mng. Eng.*, Sept. 1899.

[‡] Poggendorff, *vide* Gore's *Electrolytic Sep. of metals*, p. 57.

[§] Gore in *Proc. Roy. Soc.*, vol. xxx., p. 38.

be used to precipitate gold and silver from their solutions. Moldenhauer proposed to use aluminium as a precipitant in presence of free alkali, when the cyanide is regenerated and alumina formed, thus—

$$6AuKCy_2 + 6KOH + 2Al = 6Au + 12KCy + 3H_2O + Al_2O_3$$

It is claimed that there is comparatively little waste of aluminium.

The following table * shows the actual amounts of metals (stated in grains per square inch of surface per hour) dissolved by a solution containing 1·86 per cent. of KCy:—

	At 60° F.		At 160° F.	
	Wt. dissolved.	Amounts dissolved, Gold being taken as 1.	Wt. dissolved.	Amount dissolved, Gold being taken as 1.
Al, . .	·0095	10·5	·4940	17·5
Zn, . .	·0054	6	·0495	1·75
Cu, . .	·0053	5·9	·1258	4·5
Ag, . .	·0016	1·5	·0070	0·25
Cd, . .	·0014	1·6	·0046	0·16
Au, . .	·0009	1	·0282	1·0
Sn, . .	·0006	·66	·0042	0·15
Ni, . .	·0005	·55	·0027	0·10
Fe, . .	Trace.	...	·0010	0·035
Pt, . .	None.	...	None.	...
Pb, . .	...	...	·0031	0·11

It will be observed that the rate of dissolution of gold is increased thirty-one times, and that the "selective action" in favour of it is also increased (except as regards aluminium) by the increase of temperature. It must be noted, however, that Dr. Gore's table refers to metals in the form of sheets: it must also be observed that the solution of KCy employed was probably impure, containing much KOH. The aluminium would be dissolved by this, and the results affected generally. The position of zinc on the table is instructive. Dr. Gore has also shown that the rate of solution of gold is increased by 50 per cent. if it is placed in contact with iron, and that it is then five times more soluble than the iron.

Bagration † states that the solution of metallic gold in cyanide of potassium is facilitated by a rising temperature. Little attention was paid to this point in the early days of the cyanide process, but in 1899 Loewy stated ‡ that experiments at the May Consolidated Gold Mine in 1893 yielded 10 to 14 per cent.

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* Dr. Gore, *Proc. Roy. Soc.*, vol. xxxvii., p. 283.

† *Journal für praktische Chemie*, Leipzig, 1844, vol. xlvii., p. 367.

‡ *Journ. of the Chem. and Met. Soc. of S.A.*, April, 1899, p. 93.

higher extraction when heat was applied. The ore treated was pyritic containing 50 to 60 per cent. of free gold, which was more affected by the heating of the solutions than the gold in the pyrites. Loewy considered that solutions should be heated above 35° C. G. A. Darling had previously pointed out* that in the six "cold" months of the year in the Transvaal, from April to September, the average extraction during seven years, between 1891 and 1898, at the Robinson gold mine was 69·5 per cent., and in the "hot months," October to March, the extraction was 72·2 per cent. Von Gernet at the same time stated† that at the Worcester Cyanide Works during September, 1898, exhaust steam was passed through pipes to heat the solution, and that the residues had contained only 8 grains per ton although no comparative figures were available. On the other hand, it has been pointed out that in the "hot months" in South Africa there is more rainfall and the organic matter in the water is reduced, and the extraction might be improved from that cause, since the presence of organic matter reduces the free oxygen in the solutions.

J. S. Maclaurin found ‡ that the rate of dissolution of pure gold, in the form of plates, in potassium cyanide solution passes through a maximum when proceeding from dilute to concentrated solutions. The maximum is reached when the solution contains 0·25 per cent. of KCy. The solubility of gold is very slight in solutions containing less than 0·005 per cent., but increases rapidly as the strength rises to 0·01 per cent., when the rate of dissolution is ten times as great as in the 0·005 per cent. solution, and about half as great as that in the 0·25 per cent. The rate increases slowly as the strength rises to 0·25 per cent., and thereafter decreases much more slowly, until in 15 per cent. solutions the rate of dissolution is about equal to that in 0·01 per cent. solutions. Higher strengths show a gradual diminution in the rate of dissolution up to saturation point. Silver is also dissolved at a maximum rate in solutions containing 0·25 per cent. of cyanide, and the changes in the rate are similar to those noted above in the case of gold, the rates for silver being always about two-thirds of the corresponding rates for gold, or, roughly, in the same ratio as the atomic weights of the two metals. In both cases there is hardly any change in the rate of solubility as the strength rises from 0·1 to 0·25 per cent. From these results, it is seen that the most active solutions are now used in ordinary practice. It is remarkable that the solubility of oxygen in cyanide solutions continually decreases as the concentration increases, and it is to this fact that Maclaurin is disposed to attribute the variations in the rate of dissolution of the gold. The oxygen which unites with the cyanide, converting it into cyanate, is thereby made inactive, as the presence of cyanate of potassium has no effect on the rate of dissolution of the gold.

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* *Ibid.*, October, 1898, p. 147. † *Loc. cit.*

‡ *Journ. Chem. Soc.*, vol. lxiii. (1893), p. 731; and vol. lxvii. (1895), p. 199.

Christy, however, contends[*] that the rate of dissolution of gold in cyanide depends on two factors, the electromotive force of gold in cyanide solutions, and that of the hydroxyl ions in the solution which may be assumed to be proportional to the solubility of oxygen, and consequently decreases as the transition is made from pure water through dilute cyanide solutions to strong ones. The E.M.F. of gold continually rises as the solutions become stronger, but is never large enough to enable gold to dissolve in the absence of oxygen. In pure water the E.M.F. of gold is $-0{\cdot}72$ volt, in solutions containing about 0·043 per cent. KCy the E.M.F. becomes zero, and it then rises somewhat rapidly to $+0{\cdot}42$ volt in a solution of 6·5 per cent. KCy, and thereafter more slowly to 0·468 in a solution containing 41·7 per cent. KCy. It follows from Nernst's theory of solution pressure that solutions of gold in cyanide free from dissolved oxygen would remain undecomposed if they contained more than 0·043 per cent. KCy, but that gold would tend to precipitate spontaneously in the absence of oxygen in solutions containing less than 0·043 per cent. KCy. This requires experimental proof.

The presence of dissolved oxygen completely alters these results, and by direct experiments Christy found that solutions containing no more than 0·00065 per cent. KCy did not dissolve gold, and that for all practical purposes the cyanide of potassium solution ceases to act when its strength falls below 0·001 per cent. Above that strength, however, the solubility of gold increases rapidly.

In mill solutions, the free oxygen may be sometimes entirely removed and aëration is necessary. In long continued treatment of concentrates it may even be necessary to draw off the solution, partially dry the ore, and turn it over to expose it to the air before a fair percentage of the gold can be dissolved.

Owing to the use of the cyanide process, and owing also to the practice still adhered to of using potassium cyanide to clean the amalgamated plates, there is some danger of gold being dissolved in the stamp mill and lost. Care must be taken (when water is used over again in stamp mills) that the water is not contaminated by mixture with waste cyanide solutions. Von Gernet found that the water in the dams on the Witwatersrand contained gold in amounts varying from traces up to 12 grains per ton.[†]

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[*] "The Electromotive Force of Metals in Cyanide Solutions." *Trans. Am. Inst. Mng. Eng.*, California Meeting, September, 1899.

[†] *Chem. and Met. Soc. of S.A.*, Jan. 1899, p. 3.

Decomposition of Potassium Cyanide.—Hydrocyanic acid is one of the weakest acids known, and is expelled from its salts by all mineral acids and many organic acids. Carbonic acid decomposes potassium cyanide in presence of water thus:—

$$2KCy + CO_2 + H_2O = 2HCy + K_2CO_3$$

The smell of hydrocyanic acid, noticeable whenever KCy or its solutions are exposed to the air, is accounted for by this reaction.

In the presence of air, potassium cyanide takes up oxygen, and is converted first to cyanate and then to carbonate—

$$KCy + O = KCyO$$
$$2KCyO + 3H_2O = K_2CO_3 + CO_2 + 2NH_3$$

These reactions are much more rapid if heat is applied. Strong solutions turn brown in the air. In dilute solutions, potassium cyanide suffers hydrolytic dissociation, and is partly changed into HCy and KOH. It results from this that the passage of a stream of any neutral gas such as nitrogen through the solution causes an evolution of hydrocyanic acid, while the solution becomes alkaline. The equation is—

$$KCy + H_2O = HCy + KOH$$

The extent of the hydrolytic dissociation is 1·12 per cent. of the salt in a solution containing 0·65 per cent. KCy and 2·34 per cent. of the salt in a solution containing 0·16 per cent. KCy.* In weaker solutions, the extent of the hydrolytic dissociation is more. If the solution is boiled with acids or alkalies, hydrolysis of the cyanide occurs rapidly, ammonia and formic acid being formed, thus—

$$KCy + 2H_2O = NH_3 + HCO_2K$$

This equation does not represent the whole effect, as acetates and other organic substances are also formed. The reactions proceed slowly, even if the solution is cold and neither acids nor alkalies are present. It is obvious, from the observed facts of the decomposition of cyanide solutions mentioned above, that (1) the solution should be kept from contact with the air as far as possible, by having closely fitting lids to the storage vats, the zinc boxes, &c., and by transferring the solution from one to the other by means of iron pipes (not open launders); and (2) the solution must be kept free from acids, which can be effected by the addition of a little alkali.

Reactions in the Zinc Boxes.—It was generally believed at one time that the precipitation of gold and silver by zinc was effected by simple displacement, according to the equation—

$$2KAuCy_2 + Zn = K_2ZnCy_4 + 2Au$$

Some considerations have been adduced, however, which throw doubt on this view. It is well known that a considerable excess of free cyanide must be present to enable the precipitation to take place quickly and efficiently,* and if the solution contains less than 0·05 per cent. KCy the solution must be in contact with the zinc for an hour. Pure sheet zinc in particular is quite unable to precipitate gold in the absence of free potassium cyanide, although the filiform zinc slowly extracts the gold from the same solution, perhaps owing to the presence of lead in the zinc.† Christy suggests that the cause of this is that hydrogen is set free by the first action of the aurocyanide of potassium on the zinc, thus—

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* J. Shields, *Phil. Mag.*, vol. xxxv. (1893), p. 377.

$$2KAuCy_2 + 2Zn + 2H_2O = 2KOH + H_2 + 2ZnCy_2 + 2Au$$

This equation practically demands the replacement of the K ions in the aurocyanide by zinc—thus—

$$2(K + AuCy_2) + Zn = (Zn + 2AuCy_2) + 2K$$

followed by a re-arrangement of the molecule of zinc-gold cyanide and the replacement of the gold by another atom of zinc. This mechanism of change seems the more probable when it is remembered that the K ions in KCy itself are displaced by zinc in the presence of water.

Assuming, then, that hydrogen is set free as above, it would form an imperceptible layer of hydrogen on the surface of the sheet zinc, which would thereby become "polarised" and protected from further action, so that the reaction would soon cease. Hydrogen would also be set free on the surface of the thread zinc, but in this case the ragged edges of the shavings would assist the gas to form into bubbles and become detached, so that less "polarisation" would occur. Hydrogen is actually given off after some time has elapsed, but there are other theories to account for its formation, as noted below. It does not appear, moreover, how the presence of free KCy would prevent polarisation by hydrogen, and the prevention of the action may be due to the formation of an invisible protective layer of insoluble $ZnCy_2$ on the surface of the zinc, which would be at once dissolved if a sufficient amount of potassium cyanide were present. In the absence of KCy, the reaction proceeds—

$$4ZnCy_2 + 4KOH = 2ZnCy_2 + ZnCy_2 . 2KCy + Zn(KO)_2 + 2H_2O‡$$

This reaction is, of course, incomplete, but unless the amount of potash in the solution were large, it is clear that some insoluble zinc cyanide would remain.

The reaction is completed by the solution of $ZnCy_2$ in free potassium cyanide.

$$ZnCy_2 + 2KCy = K_2ZnCy_4$$

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* W. R. Feldtmann in *Eng. and Mng. Journ.*, August 11, 1894.

† "On the Solution and Precipitation of Gold," by Prof. S. B. Christy, *Am. Inst. Mng. Eng.*, 1896. "The precipitation of Gold by Zinc-Thread from Dilute and Foul Cyanide-Solutions," by A. James. *Ibid.*, Feb. 1897.

‡ J. S. C. Wells in *Eng. and Mng. Journ.*, December 21, 1895, p. 585.

According to Christy's formula, 1 part of zinc should precipitate about 3 parts of gold, but in practice 1 part of zinc precipitates only from 0·06 to 0·2 part of gold, the zinc being wasted by direct dissolution in potassium cyanide and by the reactions described below.

Pure zinc has a slower action on solutions of potassium cyanide than ordinary commercial zinc containing about 1 per cent. of lead, and the action of the gold-zinc couple, formed by the black deposit of gold (which may be really a compound of gold and zinc), and the unaltered zinc, is still more vigorous. This gold-zinc couple probably develops enough electromotive force to decompose water thus—

$$Zn + 2H_2O = Zn(OH)_2 + H_2$$

The positive element, zinc, is thus oxidised, and subsequently dissolved by the cyanide solution. As a matter of fact, Messrs. Butters & Clennel* observed a vigorous evolution of small bubbles, which proved to be mainly hydrogen, in the zinc boxes at the Robinson Mine. The hydrogen thus evolved doubtless carries off mechanically some hydrocyanic acid, and so the odour of the latter, noticeable above the zinc boxes, may be accounted for. The nascent hydrogen also unites directly with hydrocyanic acid, forming methylamine, thus—

$$HCN + 2H_2 = CH_2 . NH_2$$

The presence of the methylamine may in part account for the ammoniacal odour sometimes occurring above the zinc boxes, which may, however, be also caused by the hydrolytic decomposition. The hydrate of zinc, formed as shown above, dissolves at once in excess of the cyanide—

$$Zn(OH)_2 + 4KCy = K_2ZnCy_4 + 2KOH,$$

and the increase in alkalinity of the solution is thus explained.

Caustic potash, however, acts directly on the zinc to some extent forming potassium zincate, thus—

$$Zn + 2KOH = Zn(OK)_2 + H_2$$

but the zincate is, according to N. Anderson,† at once dissolved by potassium cyanide, as follows:—

$$Zn(OK_2) + 4KCy + 2H_2O = ZnCy_2 . 2KCy + 4KOH$$

In this way the solution continually tends to become more alkaline, but is neutralised by carbonic acid absorbed from the atmosphere.

By the reactions in the zinc boxes, not only is the potassium cyanide solution weakened by decomposition, as above, but a large amount of zinc is dissolved. Goyder has shown[*] that the presence of iron in contact with the zinc is injurious, checking the rate of precipitation of the gold, and increasing the waste of zinc and the tendency for the formation of $ZnCy_2$ on the zinc. Iron screws and sieves should, therefore, not be used in the precipitation boxes. The double cyanide of zinc and potassium is not available for the solution of the precious metals, and, speaking generally, as long as an excess of zinc is present, no gold will be dissolved by a solution of potassium cyanide flowing through the boxes.

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* *Eng. and Mng. Journ.*, 1892, p. 416.

† *Journ. Colo. Sci. Soc.*, April, 1895. Quoted by Ingalls. *Min. Industry for 1895*, p. 330.

An exception may be cited, however, in cases where solutions are aërated as they pass to the boxes. Caldecott stirred zinc-copper shavings with a solution of cyanide through which air was passed, with the result that gold previously deposited on the metal was rapidly dissolved.[†]

The presence of large quantities of the double cyanide of zinc and potassium in the solutions is not prejudicial to the solvent action of the simple cyanides. At the Mercur Mine the stock solution was apparently as efficacious after nine months' use as at the start, although it must have contained large quantities of zinc cyanide. Feldtmann showed that gold in ores can be dissolved by zinc-potassium cyanide, but J. S. C. Wells points out that the double cyanide remains undecomposed by gold so long as any simple cyanide is present. An accumulation of base heavy metals in the solution can be got rid of by the addition of soluble sulphides.

Ehrmann has shown[‡] by experiments on solutions of various strength, that about as much gold is precipitated by zinc in twenty-four hours at 20° C. as in two hours at 80° C. In the latter case, however, there is, according to MacArthur, "an enormous waste of cyanide by the formation of urea, which manifests itself by its strong unpleasant odour."

Action of Potassium Cyanide on Metallic Salts and Minerals Occurring in Ores.—The ordinary gangue of most ores (viz., silica and silicates of the alkalies and alkaline earths) exercises no direct influence on the cyanide solution. The carbonates of the alkaline earths are also probably without influence. The decomposing effects of sulphides of the heavy metals vary with the physical state of the sulphides.

Messrs. MacArthur and Forrest found that dilute cyanide solutions exercise a "selective action" in dissolving gold and silver in whatever form they may be present, in preference to sulphides or other salts of the base metals. There are exceptions to this rule, some of which are noted in the sequel. "For instance, cyanide of potassium solution has a strong tendency to dissolve precipitated sulphide of zinc, but its action on the natural sulphide of zinc, blende, is almost *nil*. The same holds for compounds of iron, and thus we prove selective action by the average result of a series of experiments on ores. Let us suppose a pyritic ore containing about 7 per cent. of iron and 8 per cent. of sulphur with about 1 oz. of gold to the ton. After grinding, this ore is treated with a solution containing about 1 per cent. of cyanide of potassium. The most of the gold will be dissolved and the rest of the ore left practically untouched. It is obvious that the amount of cyanogen contained in the solution is insufficient to combine with the iron present in the ore, yet, notwithstanding the much greater mass of iron sulphide present and open to attack, it is the gold that is selected for action by the cyanide solution. Taking the average result of our work we find that a higher percentage of gold than of silver is extracted, which justifies us in concluding that the selective action is greater on the former than on the latter. One of the ores on which our early investigations were done was composed as under :—

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[*] *Chem. News*, vol. lxxiii. (1896), p. 272.
[†] *Chem. and Met. Soc. of S.A.*, July, 1899, p. 147.
[‡] *Ibid.*, April 17, 1897.

Copper,	0·15 per cent.
Arsenic,	15·09 ,,
Antimony,	Traces.
Sulphur,	14·65 per cent.
Iron,	18·77 ,,
Silica,	36·20 ,,
Lead,	2·66 ,,
Zinc,	4·00 ,,
Alumina,	4·20 ,,
Gold, per ton,	2 ozs. 2 dwts. 16 grs.
Silver, ,,	2 ozs. 13 dwts. 8 grs.

In this ore we had an extraction of gold 85 per cent., silver 50 per cent., for a consumption of cyanide of about 0·45 per cent., and investigations showed that the action was directed in the order, gold, silver, iron, zinc, copper. For the amount of cyanide consumed it is obvious that the amount of base metals dissolved must have been very slight.

"The consumption of cyanide on fresh concentrates varies naturally with the composition of the concentrates. In many cases it is less than 0·2 per cent. of their weight. When the concentrates contain marcasite there is a greater consumption of cyanide than when the pyrites is entirely of the ordinary yellow cubical description. The presence of compounds of copper, physically soft, also tends to increase the consumption."

Increase of consumption with increase in strength of cyanide is shown in the following table, given by MacArthur. When one of the stronger solutions is used, the amount of cyanide consumed is equal to the whole amount present in the weaker solutions :—

	Strength of Cyanide Solution.	Consumption of Cyanide.	Extraction.	
	Per cent.	Per cent.	Au. Per cent.	Ag. Per cent.
Decomposed pyritic ore containing— Au. 1 oz. 8 dwts 3 grs. Ag. 1 „ 3 „ 3 „	0·125	0·090	93	70
	0·250	0·110	91	72
	0·500	0·150	93	73
	1·000	0·230	91	76
Pyritic ore containing— Au. 3 ozs. 11 dwts. 1 gr. Ag. 1 „ 9 „ 17 grs.	0·125	0·040	86	76
	0·250	0·050	85	74
	0·500	0·050	89	82
	1·000	0·150	88	79

It has been laid down as a general rule that oxides, hydrates, carbonates, sulphates and sulphides of those metals which are electro-positive to gold in cyanide solutions are dissolved more rapidly than the last named metal, whether it is present in the metallic form or contained in its commonly occurring salts.

This rule certainly applies to the precipitated salts commonly occurring in the laboratory, but J. S. MacArthur has shown that the case may be quite different when the naturally occurring minerals are concerned. Thus, not only is precipitated sulphide of copper rapidly dissolved, but also a sooty form of the same substance occasionally met with as a mineral occurring in ores. On the other hand, fused copper matte is scarcely acted on at all, and in a great many cases the same may be said of the hard dense sulphides of copper usually found in nature. Sulphide of zinc exhibits the same differences of behaviour: the "black-jack" concentrates of the Ravenswood Mine, Queensland, can be treated with good results, little zinc being dissolved. Again, oxide of copper, if freshly precipitated, is strongly acted on by the cyanide, but if it is heated to dull redness in a muffle it becomes insoluble, and a large excess of this material added to a gold ore makes no difference in the percentage of extraction, while the consumption of cyanide is not increased by its presence.

The action of cyanide solutions on sulphide of silver is similarly dependent almost entirely on its physical state. Experiments conducted by Fresenius showed that if a weak solution of silver nitrate is precipitated by a weak solution of sulphide of ammonium, the resulting sulphide of silver is soluble in a weak solution of cyanide of potassium. On the other hand, if strong solutions of silver nitrate and ammonium sulphide are mixed together, the precipitated silver sulphide can only be dissolved by concentrated solutions of potassic cyanide, and if this solution is subsequently diluted with water the sulphide of silver is precipitated. These results tend to show that sulphide of silver is not decomposed by cyanide of potassium, but is held in solution by it as a hydrate. Similar peculiarities in the behaviour of sulphide of silver are observed in practice when ores are being treated, and in this case an increase in the strength of the solution quickens the action of the potassium cyanide even though dilute solutions may be eventually efficacious if enough time is allowed.

A number of experiments made by Louis Janin, Jun.,* on various salts of silver point to the following conclusions:—Silver chloride is readily soluble in cyanide, and the arseniate is also rapidly dissolved. Silver sulphide and antimonide are less easily acted on, but are not so refractory as metallic (cement) silver. The presence of copper salts appears to exercise a detrimental action on the solubility of silver sulphide.

Prof. Christy's experiments on the electromotive force of metals and minerals in cyanide solutions, compared with

	OSTWALD'S NORMAL ELECTRODE = - 0·56 volt.		
	$\frac{M}{1}$ K Cy or 6·5 %	$\frac{M}{10}$ K Cy or 0·65 %	$\frac{M}{100}$ K Cy or 0·065 %
	E.M.F. in Volts.	E.M.F. in Volts.	E.M.F. in Volts.
Zinc, commercial, . .	Not given.	+0·77	+0·59
Bornite, ...	+0·45	+0·25	-0·16
Gold, pure, ...	+0·37	+0·23	+0·09
Silver, pure, . .	+0·33	+0·15	-0·05
Copper glance, . . .	+0·29 (?)	+0·25	+0·05
Lead, ...	+0·13	+0·05	+0·01
Mercury, pure, . . .	-0·09	+0·01	-0·11
Gold, amalgamated, . .	...	...	-0·13
Niccolite, ...	-0·11	-0·17	-0·44
Iron, ...	-0·17	-0·24	-0·24
Copper pyrites, . . .	-0·20	-0·34	-0·44
Pyrites, ...	-0·28	-0·42	-0·48
Galena, ...	-0·28	-0·48	-0·52
Argentite, ...	-0·28	-0·56	-0·55 (?)
Berthierite, ...	-0·30	-0·52	-0·52
Magnetic pyrites, . .	-0·30	-0·40	-0·54
Fahlore, ...	-0·36	0·52	-0·52
Mispickel, . . .	-0·40	0·45	-0·54
Cuprite, ...	-0·43	-0·55	-0·57
Electric light carbon, . .	-0·46	-0·52 (?)	-0·57
Blende, ...	-0·48	-0·52	-0·55
Boulangerite, . . .	-0·50	-0·55	-0·55
Bournonite, ...	-0·50	-0·55	-0·56
Ruby silver ore, . . .	-0·54	-0·53 (?)	-0·54
Stephanite, ...	-0·54	-0·55	-0·52 (?)
Stibnite, ...	-0·56	-0·56	-0·56

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* *Eng. and Mng. Journ.*, December 29, 1888.

Ostwald's normal electrode, show* that many minerals tend to be less rapidly dissolved than pure metallic gold. The foregoing table is abridged from the one given by him. Compare Gore's table on p. 350.

Prof. Christy observes that in testing the minerals it was difficult to get good electrical contact between the conducting platinum wires and the rough surface of the mineral fragment, so that the results are only provisional, especially as the resistance in some cases, such as zinc blende, stibnite, &c., was very high. This would tend to make the results for the minerals too low. Nevertheless, the table is interesting, and shows, for example, that copper pyrites has hardly more tendency to go into solution than pure pyrites, while bornite and copper glance have a strong tendency to set up electric currents and to dissolve. It is clear from the table that, in the case of the samples which Christy used, pure copper pyrites, galena, argentite, magnetic pyrites, fahlore, mispickel, blende, boulangerite, bournonite, ruby silver ore, stephanite, and stibnite, when free from their oxidation products, are apparently little acted on by cyanide solutions, but he gives no information as to the physical condition of the minerals tested.

Action of Potassium Cyanide on Oxidised Pyrites.—When the pyrites occur in tailings which have been subjected to the action of the weather for some time before treatment, compounds are formed which are more prejudicial to the solution than the sulphides. Sulphide of iron, FeS_2, is oxidised by air and water, ferrous sulphate and free sulphuric acid being formed, thus—

$$FeS_2 + H_2O + 7O = FeSO_4 + H_2SO_4$$

The protosulphate suffers further oxidation, and normal ferric sulphate ($Fe_2 . 3SO_4$) is produced, which eventually loses acid and becomes a soluble basic sulphate, $Fe_2O_3 . 2SO_3$. Other basic salts of complex and unknown compositions appear to be formed also.

W. A. Caldecott has investigated the products of decomposition of iron pyrites contained in slimes accumulated in dams or pits in the Transvaal, with the following results.† His view is that the main stages in the oxidation of pyrites or marcasite are as follows:—

(1) FeS_2, . . .	Iron pyrites.
(2) $FeS+S$, . .	Ferrous sulphide and sulphur.
(3) $FeSO_4+H_2SO_4$, .	Ferrous sulphate and sulphuric acid.
(4) $Fe_2(SO_4)_3$, . .	Ferric sulphate.
(5) $2Fe_2O_3 . SO_3$, .	Insoluble basic ferric sulphate.
(6) Fe_2O_3, . .	Ferric oxide.

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* See his paper in *Trans. Am. Inst. Mining Eng.*, California Meeting, Sept. 1899, p. 33.

† *Proc. Chem. Soc.*, April 29, 1897. *Proc. Chem. and Met. Soc. S. Africa*, July 17, 1897.

Whether the decomposition proceeds in this order or not, it is certain that all the products named are found in weathered tailings.

In the presence of such oxidised copper and iron pyrites, the following reactions take place:—

(1) The free sulphuric acid liberates hydrocyanic acid.

$$H_2SO_4 + 2KCy = K_2SO_4 + 2HCy$$

(2) Ferrous sulphate reacts on the cyanide, forming ferrous cyanide, which dissolves in the excess of potassium cyanide, so that it does not appear in the free state.

$$\begin{aligned} FeSO_4 + 2KCy &= FeCy_2 + K_2SO_4 \\ FeCy_2 + 4KCy &= K_4FeCy_6 \end{aligned}$$

The potassic ferrocyanide, if sufficient acid be present, reacts with fresh ferrous sulphate forming a bluish-white precipitate.

$$FeSO_4 + K_4FeCy_6 = K_2Fe_2Cy_6 + K_2SO_4$$

This precipitate oxidises in the air to Prussian blue if free acid is present—

$$4K_2Fe_2Cy_6 + O_2 + 2H_2SO_4 = 3FeCy_2 . 2Fe_2Cy_6 \text{ (Prussian blue)} + K_4FeCy_6 + 2K_2SO_4 + 2H_2O$$

Both these precipitates are decomposed by potash or soda and therefore cannot be formed in their presence. The reactions may be represented as follows:—

$$\begin{aligned} K_2Fe_2Cy_6 + 2KOH &= K_4FeCy_6 + Fe(OH)_2 \\ 3FeCy_2 . 2Fe_2Cy_6 + 12NaOH &= 3Na_4FeCy_6 + 2Fe_2(OH)_6 \end{aligned}$$

Consequently, if free acid is not present Prussian blue is hardly formed at all, as the solution soon becomes alkaline, and the precipitate is decomposed as fast as it is formed.

It follows from these reactions that if the blue colour of Prussian blue is visible in the vats or on the surface of the tailings heaps, an enormous waste of cyanide must have taken place, and the matter should be at once investigated.

(3) Ferric sulphates are decomposed by potassium cyanide, hydrocyanic acid being evolved and ferric hydrate precipitated.

(4) A mixture of ferrous and ferric sulphates produce Prussian blue by reacting with potassium cyanide, ferrocyanide of potassium being formed at first as above; the equation is—

$$3K_4FeCy_6 + 2Fe_2(SO_4)_3 = 3FeCy_2 . 2Fe_2Cy_6 + 6K_2SO_4$$

Here again the waste of cyanide is prevented by keeping the solutions alkaline.

(5) Sulphate of copper, $CuSO_4$, acts differently from $FeSO_4$, cuprous cyanide, Cu_2Cy_2, being formed, soluble in excess of KCy to $K_2Cu_2Cy_4$, a compound very prone to decomposition. Copper sulphate also gives a precipitate with potassium ferrocyanide, thus—

$$K_4FeCy_6 + CuSO_4 = K_2CuFeCy_6 + K_2SO_4$$

(6) Ferrous hydrate, when formed as above, is instantly dissolved in KCy, thus—

$$Fe(OH)_2 + 6KCy = K_4FeCy_6 + 2KOH$$

Ferric hydrate, however formed, does not act on potassium cyanide. Its only action is mechanical, as it collects in a gelatinous mass on the filters and checks the flow of liquid.

Copper and zinc in the condition of hydrates or carbonates are quickly dissolved in preference to the precious metals. If sulphates of these metals are formed in an ore containing limestone or clay, double decomposition occurs with the production of sulphate of lime or alumina, and oxides or carbonates of the heavy metals, which are dissolved by the cyanide, thus—

$$\begin{aligned} ZnSO_4 + CaCO_3 &= ZnCO_3 + CaSO_4 \\ ZnCO_3 + 2KCy &= ZnCy_2 + K_2CO_3 \end{aligned}$$

Effect of Alkalies.—Since acidity of the ore causes decomposition of the cyanide, an obvious method of reducing the loss is to add alkali in some form. Before doing this, the free sulphuric acid and soluble salts may be removed by leaching with water, and then a solution of caustic soda or lime is run on to the ore, and after standing for some time is drained off and followed by the cyanide solution. The insoluble basic salts are thus converted into ferric hydrate and soluble sulphates—

$$\begin{aligned} Fe_2O_3 . 2SO_3 + 4NaOH + OH_2 &= Fe_2(OH)_6 + 2Na_2SO_4 \\ 2Fe_2O_3 . SO_3 + 4NaOH + 4 . OH_2 &= 2Fe_2(OH)_6 + 2Na_2SO_4 \end{aligned}$$

Leaching with water then removes the excess of alkali; but, as this cannot be done completely, except with considerable expenditure of time, it is usual to use lime instead of soda. Although the action on the iron salts is slower, an excess of lime is less detrimental than soda to the cyanide solution, and does not attack the zinc. It is found that, even after treatment of the oxidised pyrites by alkalies, the loss of cyanide is much greater than in the case of free milling ores. The reason for this may, in part at least, be attributed to the action of soda on the protosalts (such as sulphates or carbonates) of copper, zinc, &c., by which these metals are precipitated as hydrates, readily soluble in KCy. The preliminary washing with water must always be carefully performed, until no coloration is obtained with ammonium sulphide, so as to remove the soluble salts as far as possible, but some always remain and are converted into hydrates by the alkali.

It is now usually regarded as more advantageous to add lime to the ore before it is charged into the vats, instead of as an alkaline solution. The necessary amount is added to each truck load of tailings, or is fed into the stamp battery with the ore, and is intimately mixed with it by the time it is charged into the vat.

The amount of alkali to be mixed with a charge of ore is determined by laboratory experiments, adding little by little an alkaline solution of known strength to a given weight of the ore, until the whole is neutral to litmus paper.

Re-Precipitation of Gold and Silver in the Leaching Vats.—If the solution is acid there is said to be some danger of a precipitation of gold previously dissolved, insoluble aurous cyanide being thrown down, according to the equation—

$$KAuCy_2 + HCl = KCl + HCy + AuCy$$

This, however, need not be feared as long as there is an excess of KCy, which must all be destroyed by the acid before the aurous cyanide can be precipitated. Moreover, aurocyanhydric acid, $HAuCy_2$, is formed and remains in solution, AuCy being precipitated only as the HCy leaves the solution by evaporation.* There is danger in transferring a solution containing gold to a vat containing pyritic material. If the latter should contain any soluble salts of the heavy metals, insoluble salts are thrown down, *e.g.*:—

$$2KAgCy_2 + ZnSO_4 = K_2SO_4 + ZnAg_2Cy_4$$

The salt $ZnAg_2Cy_4$ is probably a true double salt, but the opinion has been expressed that it is merely a mixture of simple cyanides.

A. N. Mackey attributes re-precipitation of gold to the action of free alkali on fine sulphides,† by which soluble sulphides are produced. In one case in which the protective alkali amounted to 0·04 per cent. and the cyanide to 0·02 per cent., complete re-precipitation of dissolved gold took place in 18 hours. The slimes in contact with the solution contained 0·5 per cent. of FeS_2. By reducing the alkalinity to 0·01 per cent., the re-precipitation was prevented from occurring.

Methods of Increasing the speed of Action of Potassium Cyanide on Gold.—These methods are described here for convenience, as being more intelligible after the chemistry of the process has been discussed. The necessity of the presence of oxygen has already been dwelt on above. It has, however, been frequently pointed out that in the interior of a mass of ore undergoing treatment the conditions are not favourable for the maintenance of a sufficient quantity of oxygen in a free state. Both the cyanides and the pyrites of the ore tend to unite with it, and further absorption of free oxygen from the air is extremely slow. Hence the time required for the treatment of a charge is many hours, or even days, although under favourable conditions the gold could be dissolved in a few minutes, or at most in two or three hours. To supply the oxygen, various oxidising substances have been tried, such as the passage of a current of air through the solution, and the addition of various materials (see above, p. 349), some of which were tried by Dixon in 1877 * in his unsuccessful attempt to find a process for treating refractory ores. These substances hasten the solvent action of cyanides on metallic gold, and have sometimes been used in practice, especially in the United States, although they act as "cyanicides," destroying some of the solvent by direct or indirect oxidising effects.

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* See *Bull. Soc. Chim. de Paris*, May 5, 1878, p. 422.

† "Re-precipitation of Gold from Cyanide Solutions," *Inst. of Mng. and Met.*, May, 1905.

Dr. N. S. Keith suggests † that the action of oxygen is due to its strong electro-negative relation to gold in cyanide solutions, and has experimented with various materials which are electro-negative to gold in solutions of cyanide of potassium. In the list of such materials, given above, are carbon, iron, lead, and mercury (see pp. 350 and 360). Keith tried the effect of finely-powdered carbon mixed with the ore, and continually agitated with it in a cyanide solution. He found that the gold was more rapidly dissolved than if no carbon had been present, and supposed that some of the particles of gold came into contact, with the carbon and formed galvanic couples, in which the electro-positive element, gold, was quickly acted on. Such contact, however, could not be attained in practice except to a small extent, and the method is therefore useless, whilst the use of lead or iron in this way is *a fortiori* impossible. On the other hand, mercury can be more readily subdivided and distributed through the ore, but, as it amalgamates with the gold, the conditions are changed, and, as a matter of fact, the gold in pasty amalgam is only slightly more rapidly dissolved by cyanide than is pure gold.

From the fact that mercury is electro-negative to gold in cyanide solutions, Skey in 1876 ‡ concluded that metallic gold in contact with a solution of mercury cyanide would rapidly dissolve and mercury be reduced. He found this to be the case alike with gold and silver, which dissolved with almost equal readiness.

Keith Process.—In 1895, Keith proposed § to add a small quantity (2 ozs. to 12 ozs. per ton of liquid) of potassium mercuric cyanide, $HgCy_2 . 2KCy$, to ordinary cyanide solutions to quicken their action by enabling the presence of free oxygen to be dispensed with. ‖ The gold displaces the mercury from solution, and so is dissolved, whilst the mercury is precipitated on the surface of the particles of gold and forms amalgams.

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* *Chemical News*, Dec. 20, 1878, p. 293.

† *Engineering*, vol. lix. (1895), p. 379.

‡ *Transactions and Proceedings of the New Zealand Institute*, vol viii., p. 334.

§ *Engineering*, *loc. cit.*

‖ A similar proposition had been made and patented by J. S. MacArthur some years previously.

If, for example, as was pointed out by Dr. Hood, gold is digested with an aqueous solution of potassium cyanide and one or other of the chlorides of mercury, the action is represented, so far as weights are concerned, by the following equations:—

$$2Au + HgCl_2 = Hg + 2AuCl$$
$$2Au + Hg_2Cl_2 = Hg_2 + 2AuCl$$

These equations do not, of course, represent the whole of the interchanges. The presence of the alkaline cyanide evidently assists in dissolving the gold, for a solution of $HgCy_2$ alone in water has no action on gold. By using an excess of gold, the whole of the mercury can be removed from solution, and equivalents of gold dissolved as represented by the equations given above, 2 × 196·8 parts of gold being dissolved when 200 parts of mercury are added as mercuric chloride or 2 × 200 parts as mercurous chloride. In some experiments made by the author employing 0·5 gramme mercuric chloride and 1·0 gramme potassium cyanide, the amount of gold dissolved by the hot, strong (5 per cent.) solution in thirty minutes was 0·724 gramme, whilst theory requires 0·726 gramme. Under similar conditions, the amount of gold dissolved by potassium cyanide alone was only 0·010 gramme.

Keith supposed that the gold would thus be rapidly dissolved, and the precipitated mercury would then be re-dissolved in the cyanide, and thus be ready to react as before. This view seems to be incorrect. According to the author's experiments, the gold is at first rapidly dissolved, and the mercury precipitated. As the action proceeds, however, the dissolution of the gold becomes slower and slower, the mercury appearing to protect it more and more as the percentage of gold in the amalgam is diminished. If the particles of gold are only moderately fine (*e.g.*, gold precipitated from the solution of the chloride by sulphurous acid) the action becomes extremely slow after about 85 per cent. of the gold has been dissolved, the amalgam then consisting of about three parts of mercury to one of gold. If, on the other hand, very finely-divided gold is used, such as gold leaf, the action is fairly rapid until about 95 per cent. of the gold is dissolved, and in one case 98 per cent. of such gold was dissolved in four days by a solution containing 1·5 per cent. of $HgCy_2 . 2KCy$. In the case of gold leaf, however, which contains both silver and copper, the rate of dissolution is higher than it would be for pure gold in a similar state of subdivision, as the presence in the alloy of either silver or copper favours the dissolution. The retarding effect exercised by metallic mercury when amalgamated with the gold is exemplified by the results of some experiments in which the solution contained 1·5 per cent. of $HgCy_2 . 2KCy$, the time of treatment was thirty-six hours, and the weights and states of aggregation of the metal treated were approximately the same. Under these conditions, about 86 per cent. by weight of some samples of pure gold were dissolved, and only 14 per cent. of the gold contained in amalgams consisting of two parts of mercury and one of gold. Dissolution of gold by solutions containing mercury cyanide is greatly expedited by heat. In the author's experiments on ores, the quickening effect of mercury cyanide on solutions of cyanide was very slight. The use of bromine in the Sulman-Teed process has been already described (p. 332). Sodium peroxide has also been tried, but it does not seem to be used in practice.

LABORATORY WORK IN A CYANIDE MILL.

Testing Cyanide Solutions.—The ordinary method of estimating the amount of potassium cyanide present in a liquid is by titration with a standard solution of silver nitrate. Silver cyanide is formed, and re-dissolves in the excess of potassium cyanide until one-half of the latter has been decomposed. The equations are as follows:—

$$AgNO_3 + KCy = AgCy + KNO_3$$
$$AgCy + KCy = KAgCy_2$$

When one-half of the KCy present has been converted to AgCy, an additional drop of $AgNO_3$ solution causes the formation of a permanent white precipitate of AgCy. The amount of silver solution added is then read off, and the percentage of cyanide calculated. The equation of the end reaction is—

$$KAgCy_2 + AgNO_3 = 2AgCy + KNO_3$$

A few drops of a solution of potassium iodide are often added, in accordance with a suggestion made by J. S. MacArthur, to make the end reaction sharper, and to prevent inaccuracy through the presence of ammonia or other substances in which silver chloride is soluble.

The presence of soluble sulphides prevents the test from giving any result, owing to the formation of brown sulphide of silver on the addition of the first drop of silver nitrate. In this case, the sulphides are precipitated by shaking with freshly precipitated carbonate of lead, before titrating.

Methods of Testing Cyanide Solutions containing Zinc.—The method given above is difficult to apply when solutions containing soluble cyanides of zinc and other metals require to be titrated. "A white flocculent precipitate occurs at a certain stage, probably consisting of simple (insoluble) cyanide of zinc, formed by decomposition of the soluble double cyanide—

$$K_2ZnCy_4 + AgNO_3 = KAgCy_2 + ZnCy_2 + KNO_3$$

This precipitation occurs long before the whole amount of potassium has been converted into the soluble double salt of silver ($KAgCy_2$), for the solution after the appearance of the flocculent precipitate, still gives the Prussian blue precipitate with acidulated ferrous sulphate."[*] Bettel suggested[†] the addition of an excess of caustic soda, and the iodide indicator, titrating with silver nitrate. This gives the sum of the free cyanide, hydrocyanic acid, and part of the double cyanides. Part of the zinc potassium cyanide is decomposed by the alkali, sodium and potassium zincates and free cyanide being formed.[‡] The greater the excess of alkali the more complete the reaction would be. Compound cyanides, such as ferrocyanides, sulphocyanides, &c., are not decomposed by caustic soda, so that their cyanogen is not included in the estimation. Owing to the action of mass, none of these actions are absolute, and the estimation is only approximate. The determination does not appear to be of much practical value.

G. A. Goyder (*Chem. News*, Aug. 16, 1895, p. 80) found that the amount of free cyanide shown by this method varies with the dilution. Thus a sump solution from the Mount Torrens Works, Australia, was found by titration to contain 0·04 per cent. KCy. On adding an equal bulk of water, it appeared to contain 0·05 per cent. KCy (instead of 0·02 per cent.), and on diluting with two volumes of water, it showed 0·04 per cent. KCy instead of 0·013 per cent. Goyder also observed that a solution of 1 per cent. of crystallised double cyanide of zinc and potassium appeared to have one-thirtieth of its cyanogen as free cyanide, and on diluting to 250 volumes (0·004 per cent. K_2ZnCy_4), the whole of the cyanogen appeared to be in the state of simple cyanide. The potassium cyanide set free by the action of the alkali is doubtless available, both to dissolve gold and to keep silver cyanide in solution. In any case, it seems quite likely that the free cyanide shown by simple titration represents the cyanide available for the dissolution of gold, provided that the alkalinity of the mill solution is made equal to that of the solution when it is tested in the laboratory.

L. M. Green proposes[§] the following methods of analysis in order to determine—

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[*] *Eng. and Mng. Journ.*, 1892, p. 417.

[†] *Chem. News*, 1895, vol. lxxii., p. 287. See also "Notes on the Analysis of Cyanide Solution," by J. E. Clennel, *Proc. Chem. and Met. Soc. of S. Africa*, Dec. 21, 1895, and *The Chemistry of Cyanide Solutions*, by J. E. Clennel, London and New York, 1904, pp. 34-39.

[‡] J. E. Clennel (*Chemistry of Cyanide Solutions*, p. 36) is doubtful whether free potassium cyanide and zincate of potash can coexist in the same solution. He has not, however, considered the effect of mass-action.

[§] Paper read at the *Inst. of Mng. and Met.*, Oct. 17, 1901.

(1) The total cyanide.
(2) The total amount of "protective alkali"—that is, the free alkalies and alkaline earths, together with half the monocarbonates.
(3) The ferrocyanides.
(4) The zinc.

(1) Take 50 c.c. of the solution. Add excess of NaHO and a little KI solution. Titrate with $AgNO_3$ till there is a distinct permanent yellowish cloudiness.

Result = T c.c. $AgNO_3$

(2) Take 50 c.c. of the solution. Add excess of K_4FeCy_6 solution. Then run in 2 T c.c. $AgNO_3$. Add phenolphthalein and titrate with $\frac{N}{10}$ nitric acid till colourless.

Result = p c.c. $\frac{N}{10}$ HNO_3

If an excess of barium chloride is added before this test is made, the result gives the alkalies and alkaline earths only.

(3) Take 50 c.c. of the solution. Add p c.c. $\frac{N}{10}$ nitric acid: then add 2 T c.c. $AgNO_3$, one drop of K_2CrO_4, and continue adding $AgNO_3$ until there is a permanent red coloration.

Result = n c.c. $AgNO_3$. (Total amount added.)

(4) Take 50 c.c. of solution. Add 10 c.c. $\frac{N}{10}$ Na_2CO_3. Run in 2 T c.c. $AgNO_3$. Shake well. Add phenolphthalein, and then add $\frac{N}{10}$ HNO_3 till colourless. Add 1 c.c. more acid, shake up, and add $\frac{N}{10}$ Na_2CO_3 till the clear solution is faintly pink. Add excess of K_4FeCy_6, and titrate the strongly coloured solution with $\frac{N}{10}$ HNO_3 till colourless.

Result = s c.c.

(5) Repeat No. 4, but adding n c.c. $AgNO_3$, and shaking, instead of 2 T c.c.

Result = t c.c.

If the standard solution of $AgNO_3$ contains 13·05 grammes per litre, then the following are the factors:—

Total cyanide estimated as KCy = T × 0·02 per cent.
Protective alkali estimated as KOH = p × 0·0112 ,,
Ferrocyanides, estimated as $K_4FeCy_6 . 3H_2O$ = $(t - s)$ × 0·0351 ,,
Zinc = t × 0·0086 ,,

According to Watts[*] and Fresenius[†] the total amount of cyanogen in a solution, whether present as simple or double cyanides, may be estimated by boiling with an excess of oxide of mercury and water, when all the cyanogen is obtained as cyanide of mercury and the metals pass into oxides. The cyanide of mercury is then precipitated by nitrate of silver, with the precautions recommended by H. Rose and Finkener.[‡]

Estimation of Available Cyanide.—One of the most important points which the operator really wants to know is the dissolving power on gold of the solution. No method of estimation has been described which enables this to be readily determined. After discussing the matter, Clennel concludes[§] that the best way to determine the available cyanide is to compare the rate of dissolution of freshly precipitated gold in a standard cyanide solution with that in the solution to be tested.

Estimation of Gold and Other Metals in Cyanide Solutions.—The amount of gold in the solution is usually determined by evaporating a known bulk to dryness in lead basins, or in porcelain basins with the addition of litharge mixed with the liquid before evaporation is commenced. The determination is finished by reducing the lead by fusion in a crucible with charcoal, and cupelling the lead button. Gold can also be determined as follows:—Add an excess of a solution of $AgNO_3$, filter, reduce the precipitate with zinc and hydrochloric acid; filter, dry, cupel and part the button.[||]

The amount of zinc and other heavy metals present may be determined by concentrating the solution by evaporation, adding an excess of sulphuric acid, and heating until "almost all the sulphuric acid has been expelled. The residual mass is then free from cyanogen. It is dissolved in water, if necessary with the addition of hydrochloric acid, and the oxides determined by the usual methods. This way is not adapted for cyanide of mercury, as a little of the metal would escape with the fumes of the sulphuric acid." J. E. Clennel[¶] decomposes the cyanogen compounds by boiling with hydrochloric acid and chlorate of potash. The metals are then determined by the ordinary methods of chemical analysis.

Detection and Estimation of Sulphides in Potassium Cyanide.—In view of the marked effect produced by soluble sulphides in checking the solution of the gold, it is sometimes desirable to test for them in samples of cyanide, although it is seldom necessary to make exact determinations

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[*] *Dict. of Chem.*, 1864, vol. i., p. 202.

[†] *Quant. Chem. Anal.*, 7th edition, 1876, vol. i., p. 376.

[‡] *Loc. cit.*

[§] *Chemistry of Cyanide Solutions*, p. 56.

[||] See also p. 346.

[¶] *Eng. and Mng. Journ.*, June 29, 1905, p. 1230.

The most delicate test is to add a solution of nitro-prusside to the cyanide solution. The most minute trace of alkaline sulphide is shown by the appearance of a brilliant purple colour. Nitroprussides are formed by adding nitric acid to a solution of ferrocyanide of potassium.

Dr. Loewy's method for the estimation of sulphides in cyanide solutions is to add nitro-prusside of soda, which gives the violet colour. He dissolves 2 grammes of the sample of cyanide to be tested in 100 c.c. water, and compares it with a solution of cyanide of similar strength free from sulphides. He then adds 1 c.c. of a solution containing 5 per cent. of sodium nitro-prusside and a very little cyanide to each solution, and then adds a solution of sulphide of soda of known strength to the pure cyanide until the violet colour is of the same depth in each. The calculation of the result is then simple. The method is applicable to solutions in which the amount of sulphur lies between 0·0005 and 0·0015 per cent. The solutions are adjusted to bring the sulphur within these limits.

Estimation of the Reducing Power of Cyanide Solutions.—This is more important than the estimation of sulphides, as the solvent action of the solution depends on its oxygenating power, or the reverse. The methods adopted are to add an excess of sulphuric acid, and either to titrate with potassium permanganate (Clennel's method), or to add an excess of permanganate and titrate back with sulphocyanide (Bettel's method). In the first case the end point is denoted by the first appearance of a permanent pink colour, and in the second case by its final disappearance.

Estimation of Oxygen in Working Cyanide Solutions.—It is of great importance to be able to determine the proportion of free or available oxygen in mill solutions. One method, devised by A. F. Crosse,* is based on Thresh's method of estimating oxygen dissolved in water.†

Another method, proposed by Prof. Prister,‡ consists in boiling off the air contained in a solution, collecting and measuring the gases in Lunge's nitrometer, and removing the oxygen by means of a strong solution of pyrogallate of potassium made by dissolving pyrogallic acid in caustic potash. The residual gas (nitrogen, &c.) is again measured, and the difference consists of oxygen.

By Crosse's method the amount of oxygen per litre in certain cyanide solutions was found to be as follows:—Solution from Siemens-Halske process before precipitation, 4·65 to 4·69 mgrms.; tap-water, 7·7 mgrms.; the same tap-water, with 0·2 per cent.

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* *Chem. and Met. Soc. of S.A.*, August, 1898, p. 107, and Sept. 1898, p. 125; *Journ. Soc. Chem. Ind.*, Nov. 30, 1898, p. 1074.

† *Journ. Chem. Soc.*, vol. lvii. (1890), p. 185.

‡ *Chem. Met. and Mng. Soc. of S.A.*, April, 1904, p. 364.

KCy and a little ferrocyanide, 7·6 mgrms.; solution as pumped on to a leaching vat, 6·3 mgrms.; the same solution as run from the vat thirty hours later, 0·6 mgrm.; and the same from the end of the zinc boxes, 0·3 mgrm.

Estimation of Bromo-cyanogen.—An excess of hydrochloric acid is added, and this is followed by the addition of an excess of potassium iodide. Iodine is liberated, the quantities being those shown in the following equation:—

$$BrCN + HCl + 2KI = HCN + KBr + KCl + I_2$$

The iodine is estimated by titration with sodium thiosulphate in the usual way.

Testing of Ores.*—Experiments with small quantities of material in the laboratory will usually determine the maximum extraction that can be looked for. The weight of ore taken may be from 100 to 200 grammes. It is placed in a funnel or lamp glass, fitted with a perforated cork and filter bed, and washed with water, or, perhaps better still, shaken with distilled water in stoppered bottles and filtered. The filtrate may contain soluble salts of iron, copper, &c., and tests are applied for these and also for acidity, and for soluble sulphates and chlorides. The water filtrate may also be mixed with cyanide solution to determine how much cyanide, if any, is decomposed by it. In this way the value of the *water wash* on the particular ore is determined.

Preliminary tests as to the amount of alkali which it is necessary to add, and the amount of cyanide decomposed are made as follows:—(*a*) *Alkali.*—200 grammes of washed ore are made into a paste with water in a porcelain basin, and tested with litmus paper. If an acid reaction is observed, a titrated solution of caustic soda is run in, little by little, until the mixture is neutral. It is convenient to make the alkaline solution of such a strength that each c.c. corresponds to one pound of caustic soda per ton of the ore which is being tested. (*b*) *Decomposition of Cyanide.*—100 grammes of the ore are shaken in an 8-oz. stoppered bottle with 50 to 100 c.c. of a solution of KCy for fifteen minutes, then left undisturbed for twelve hours, and finally shaken again for five minutes. The available cyanide in the separated solution is then estimated and compared with that in the original solution. By merely shaking for a few minutes, most of the decomposition of cyanide is made to take place, but not the whole.

Cyanide solution of 0·1 or 0·2 per cent. is then passed slowly through the ore drop by drop, or shaken and left in contact with it for some time, and then drawn off and the filtrate tested for soluble sulphides, ferrocyanides, sulphocyanides, copper, zinc, &c., and for its reducing power. The tests recommended by Clennel are—

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* See article by J. E. Clennel in *Chem. and Met. Soc. of S.A.*, Dec. 1898, p. 180.

(1) Add hydrochloric acid. A white turbidity may indicate the presence of soluble sulphides or of zinc.

(2) After adding the acid in slight excess, add a few drops of K_4FeCy_6. A brown coloured precipitate indicates copper, a white flocculent precipitate may indicate zinc.

(3) Add lead tartrate dissolved in excess of NaOH; a brown or black colour indicates soluble sulphides.

(4) Acidulate with H_2SO_4 and add dilute $K_2Mn_2O_8$ drop by drop. If it is decolorised, *reducing power* of the solution is indicated, and aëration may be necessary.

The *percentage of extraction by cyanide* may then be tried by passing solutions through the ore, washing thoroughly by filtration, and assaying the residues to find what quantities of the precious metals have been removed, checking the results by recovering the gold from the filtrate.

The *consumption of cyanide* may be determined by titrating the solution before and after use. In this way, the maximum extraction of gold and silver from an ore which has been crushed to different degrees of fineness, with solutions at different strengths, acting at various temperatures and for different lengths of time, may be determined. The best *proportions of solution to ore* may also be determined. Such data will be of great value in determining the degree of suitability of the process to any given ore, but the results obtained cannot usually be repeated on the large scale. Mechanical difficulties preclude fine crushing beyond a certain point; the relative amount of solution used in practice must be less than the large quantities giving the best results on a small scale. Moreover, the washing and filtering is more perfectly performed in the laboratory than in the mill, but, on the other hand, the consumption of cyanide may be less in the latter case than in the former.

Clennel recommends tests on ore crushed to 10, 20, 30, 40, and 60-mesh size, with solutions of 0·01, 0·05, 0·1, 0·2, and 0·5 per cent. of KCy, amounting to 1, 1¼, 1½, 2, 3, and 4 A.T. of solution per A.T. of ore in successive washes, remaining in contact with the ore for 1, 2, 3, 4, 7, and 14 days.

Treatment in case of Cyanide Poisoning.—A coating of oil or kerosene (or rubber gloves) protects the hands from the action of the cyanide solution.

If cyanide solution is swallowed, anything that causes vomiting may be administered, but the action of cyanide is extremely rapid and every second is of value. Freshly precipitated ferrous hydrate, made by mixing oxide of magnesia, caustic potash, and ferrous sulphate, is a useful antidote. The use of nitrate of cobalt has also been suggested, and copious subcutaneous injections of hydrogen peroxide are recommended.

The patient may also be rubbed with camphor and alcohol, or cold water may be dashed upon the skin.

If hydrocyanic acid has been inhaled, an inhalation of ammonia, or chlorine, or ether, should be administered, but respirators should be worn in cases, such as in the acid treatment of gold slimes, where evolution of poisonous gases are probable. Park records the warning that in the acid treatment of gold slimes from arsenical ores, an evolution of arseniuretted hydrogen may occur, and the inhalation of this gas is very deadly, no antidote being known.

Effect of Thiocyanates on Metallic Gold.—This has been investigated by H. A. White,* who found that potassium thiocyanate, though inactive by itself, would dissolve gold if mixed with various oxidising agents, such as potassium permanganate, ferricyanide of potassium, dilute nitric acid, &c. Ferric thiocyanate also dissolved gold, but in this case it was supposed that the presence of dissolved oxygen was necessary. It is considered that the Stark process, in which thiocyanates are used, may be of value in the treatment of old tailings heaps.

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* *Journ. Chem. Met. and Mng. Soc. of S.A.*, October, 1905, p. 109.

CHAPTER XVIII.

THE REFINING AND PARTING OF GOLD BULLION.

General Considerations.—By whatever process gold may have been extracted from its ores, it is necessary to melt the crude bullion and cast it into bars so that its value may be ascertained, and that it may be put into a form convenient for transportation and sale. The name "bullion" may be conveniently restricted to the precious metals, refined or unrefined, in bars, ingots, or any other uncoined condition, whether contaminated by admixture with base metals or not. It is, however, often applied to coin, and the appellation "base-bullion" is given to the pig-lead or to copper bottoms or pig-copper, which have been obtained in smelting operations, or as the result of melting worn-out amalgamated copper plates. Such materials may only contain a few parts per thousand of gold and silver, the main portion consisting of base metals. The treatment of base-bullion, however, properly belongs to metallurgy of argentiferous lead and copper, and the descriptions given in this chapter apply to bullion which is valuable almost entirely on account of the gold and silver contained in it.

The operations to which the retorted metal, gold precipitate or bars from the chlorination mills, &c., are subjected may be summarised as follows :—

1. The bullion is melted in crucibles (a rough refining operation being usually effected at the same time) and cast in ingot-moulds.
2. Assay-pieces are cut from the cast ingots or dipped from the molten metal before pouring, and assays are made on these, by which the value and composition of the bars are ascertained.
3. The bars are then usually sold to the refineries, where the base metals are eliminated and the gold and silver separated by "parting," and cast into bars separately. Both before and after the parting it is sometimes necessary to subject the bullion to further refining operations. The bars of gold and silver thus obtained, being of a high degree of purity, are in a condition to be used for minting, or for the various industrial purposes to which they are applied.

Rough unrefined gold is frequently sold to the refineries attached to the American and Australian mints, in the state of retorted metal, &c., without being previously melted and assayed, the producing mills relying on the good faith of the officials at these establishments.

Composition of Bullion.—Bullion varies greatly in composition, and gold may be present in any proportion up to nearly 100 per cent. The gold obtained in some chlorination mills is of a high degree of purity and rarely contains much silver. This precipitated gold, however, generally makes brittle bars owing to the presence of a few parts per thousand of lead, bismuth, antimony, and other metals of high atomic volume. From some chlorination mills the gold is far from pure, owing to various causes, which include lack of care. If ferrous sulphate is used as the precipitant, the precipitate may contain large quantities of ferric hydrate from which some iron is reduced in the crucible, and if sulphuretted hydrogen is used and the gold precipitated as sulphide, it is contaminated with all the heavy metals contained in the solution, copper, iron, and lead being most often encountered. These may amount to several per cent. Retorted metal is of very different degrees of fineness, according to the nature of the ore and the course of treatment. Placer gold is usually finer than that derived from lodes, containing a smaller percentage of silver, while the nature of the material treated and the methods used in placer operations are not favourable to the contamination of the bullion with base metals, which vary in amount only from 0 to 20 parts per thousand, and seldom approach the latter figure.

Battery-retorted gold is usually much less pure than placer gold, the percentage of base metals being in particular much higher, a state of things due in great part to the difference in the method of treatment. The bullion from pan-amalgamation is less fine than battery gold, containing less gold, more silver, and more base metals. Retorted gold often contains large quantities of iron, and copper is also a common impurity.

Bullion from the cyanide process is sometimes of very low standard, containing as little as 10 per cent. of gold. The predominating impurities are zinc and lead.

The Melting Furnace.—The furnace used for melting the bullion is of simple construction. It may be round or square, with walls consisting of an outer layer of ordinary brick and an inner layer, at least 4 inches thick, of the best firebrick. There is often a complete outer casing of iron, which is useful in keeping the furnace from falling to pieces, but radiates more heat than the bricks. The fire-box in a small furnace may be about 1 foot square and about 2 feet deep; below it is an ashpit, usually lined with a cast-iron tray and provided with a working iron door, through which the air-supply of the furnace passes, and by which it is regulated. The largest furnaces at the London Mint are round, 21½ inches in diameter and 31½ inches deep. Still larger furnaces are in use at the Rand. The fire-bars are movable, and their ends rest loosely on iron supports. The top of the furnace may be made flat or sloping up towards the back at an angle of about 30°. In this case a wide flat ledge should be provided at the front, on which crucibles and moulds can rest. The top is always made of a cast-iron flanged plate, with an opening of the same area as the fire-box. This opening is closed by a cast-iron sliding door made in one or two pieces, and preferably lined with firebrick and running on rollers. The flue is placed at the back of the fire-box near the top; in a small 12-inch square furnace the cross-section of the flue should have an area of about 16 or 18 square inches—6 by 3 inches, and 4 inches square are both convenient sizes. The round furnaces at the Mint have a flue of 6 inches square. The flue communicates with a stack, which must be of brick for a distance of 2 or 3 feet from the furnace, but may be of wrought-iron tubing in its upper part. The height of the stack will depend on the position of the furnace, and should be as great as possible, 60 feet giving better results than any less amount. It has been stated that a height of 30 feet is the minimum that can be allowed in order to ensure a good draught, but very satisfactory results can be obtained with a stack only 16 feet high. The furnace can be built by any bricklayer acting under directions. No mortar is used in its construction; clay, mixed with an equal bulk of sand, being substituted for it. A sliding damper in the flue at a convenient height above the ground is necessary, so as to regulate the draught. The fuel used in such a furnace may be anthracite, charcoal, or good coke, made in coke ovens,

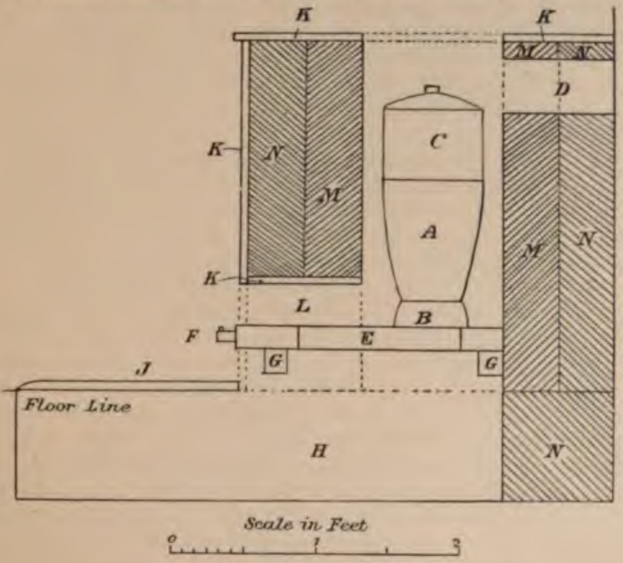

Fig. 81.—Melting Furnace.

and broken into pieces of moderate size. If the coke is of high quality, it is the most satisfactory fuel, making a hot fire and lasting for a long time, so that it does not require very frequent replenishing. Neither dust, nor very small, nor very large pieces must be used. Charcoal is preferred in the United States Mints for small charges, and anthracite for large ones.

Gas furnaces are also used, the crucible being heated by three or more burners, each with a blast of air, placed at regular intervals in the circumference. Gasolene or naphtha furnaces are also in use.

The flues should pass into dust chambers. At the San Francisco Mint the dust chamber is 12 feet high, 1 foot wide, and of the same length as the row of furnaces, and at the Melbourne Mint the dust chamber is 8 feet high and 5 feet wide. Baffle plates should be inserted in the dust chambers, which are built of brick.

Royal Mint Furnace.—One of the gold melting furnaces in use at the Royal Mint, London, is shown in sectional elevation in Fig. 81, and a fire-bar belonging to the furnace is shown in Fig. 82. The crucible A stands on B, which consists of the lower part of an old plumbago pot cut off about 2 inches from the bottom. C is the "muffle," a plumbago cylinder 6 inches high, resting on the crucible. It enables a deeper bed of coke to be used, and also gives space for charging-in bulky material. The furnace is 12 inches square and 2 feet deep above the fire-bars. The pot is 8⅝ inches diameter at the widest part, and its charge is 1,200 ozs. of standard gold. The flue D is 4½ inches deep and 5 inches wide, and communicates with a stack 45 feet high. The fire-bars, E, are 22½ inches long and 2 inches deep. They are tapered downwards, especially in the middle portion. The section of the thick part near the ends is shown in O, Fig. 82, and that of the middle part in P, Fig. 82. The projection F is used for withdrawing the bars. G G, Fig. 81, are the supports for the bars; they are of 2-inch square iron, and let into the brickwork. H is the ashpit, 12 inches by 3 feet 4 inches and 9 inches deep below the floor. It is lined with ¾-inch cast-iron plates. J is a sliding iron plate covering the exposed part of H. K K are cast-iron plates ¾ inch thick covering the furnace. There are four fire-bars and their thickened ends nearly touch, occupying 11 inches in the width of 12 inches. The spaces between the middle parts of the bars, however, are over ½ inch wide, and some air passes in between the bars, but most of the air enters the furnace through the aperture L which is 3½ inches

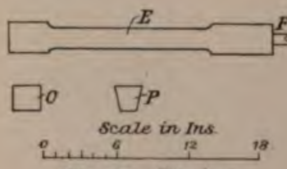

Fig. 82.—Fire-bar.

high above the fire-bars and 12 inches wide. The pot-support B rests on the two middle fire-bars, so that the outer fire-bars can be withdrawn without disturbing the pot. The furnace is lined with firebrick, M M, 4½ inches thick, the outer layers N being filled up with ordinary brick. The top of the furnace is 29 inches above the floor line. A row of eight furnaces have one stack in common. The distance between the centres of two adjacent furnaces is 2 feet 6 inches.

The Crucibles.—The bullion is melted in either graphite or clay crucibles, or in graphite pots lined with clay. Detachable clay liners are sometimes used, or a clay pot may be fitted inside a graphite guard pot. The size of the crucibles and the weight of the charges of bullion vary greatly, but in extraction mills, as a general rule, a gold-charge does not exceed 400 ozs., and a silver-charge 1,200 ozs. in weight. In mints and refineries, much larger crucibles are employed, holding different amounts up to 6,000 ozs. of metal. On the Rand No. 400 pots are used in certain cases, capable of holding 12,000 ozs. of metal.

Melting the Bullion.—All crucibles must be thoroughly annealed before being used; otherwise, the contained moisture being suddenly converted into steam when the crucible is heated rapidly, the pots crack. The crucible is kept on a shelf near the flue, for as many days or weeks as convenient, before being used. It is then placed on the top of the furnace or in the ashpit for a few hours, when it will probably be safe to hold it over the open furnace by means of the crucible tongs, until it becomes gradually warm. After a few minutes, the crucible being turned round at intervals, it can be lowered rim downwards upon the burning fuel, and as soon as the rim becomes red-hot, the crucible is quite safe, and may be turned over and placed in position for the reception of the gold. With Salamander crucibles, a less degree of care in annealing will suffice, as they are well annealed before being sold. The crucible rests on a firebrick about 3 inches thick, which is laid on the bars of the grate. If the firebrick were omitted, the bottom of the pot, resting directly on the fire-bars, would be too cold, while a layer of fuel, if placed below the pot, would soon be burnt out, and could not readily be replaced, so that the pot would sink down to the bars. The fuel is built up round the pot until it reaches to its rim, or the top of the muffle, and the fire urged until the whole pot is at a full red heat. Borax is then thrown into the crucible by means of a scoop to slag off metallic oxides and so assist the metal to melt. As soon as the borax is melted, the introduction of the bullion is commenced. The safest way to do this is to use the shoot shown in Fig. 83, which is held in position, its lower edge being inside the crucible, with the left hand, while the metal is transferred to it in a scoop by the right hand. In this way the melter avoids all danger of loss which might be encountered if the metal scrap were wrapped in paper and added by the tongs. Large pieces of metal are added by the crucible tongs. The cover, which must also have been previously well annealed, is kept on the crucible as much as possible, and the fuel pushed down with the poker to avoid scaffolding, and fresh pieces of coke added when required. The crucible is not allowed to become more than two-thirds full at any time, but more metal is added when the first supply has been melted down, and the operation repeated until the pot is sufficiently full of molten material.

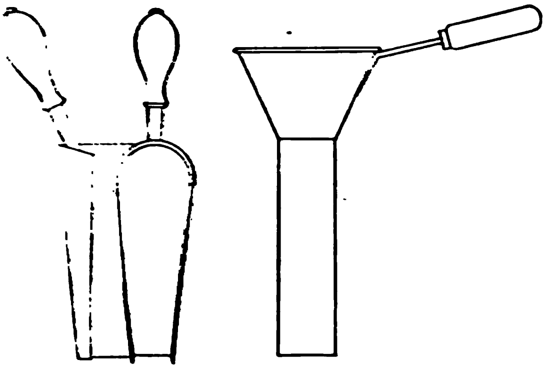

Fig. 83.

REFINING.*

Methods of Refining.—The processes in use are as follows:—

1. Volatilisation.
2. Oxidation (*a*) by air blowing or roasting.
   (*b*) by "bessemerising."
   (*c*) by nitre.
   (*d*) by metallic oxides.
   (*e*) by cupellation.
3. Chlorination.
4. Sulphurisation.
5. The use of iron or carbon.

The method to be used depends partly on the composition of the bullion, and partly on the means at the disposal of the operator. Cupellation belongs properly to the metallurgy of silver and lead, and need not be dealt with here. (See, however, Tavener's process, p. 303). The other operations, except roasting, are usually carried out in crucibles, although reverberatory or tilting furnaces are sometimes used for work on a large scale.

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* The word "refining" is used to denote the removal of impurities from gold and silver. In refineries this is called "toughening," the word "refining" being used to denote the separation of gold from silver and copper, which is called "parting" in this book.

1. **Refining by Volatilisation.**—The volatile metals are partly removed by melting the bullion. Mercury is almost all volatilised when the gold is melted. Mercury boils at 354°, sulphur at 440°, arsenic at 550°, selenium at 680°, cadmium at 770°, zinc at 940°, and tellurium at about 1,000°. Pure gold melts at 1,064°, but its melting point is lowered by the presence of impurities, and the temperature of molten bullion in an ordinary furnace may be taken as between 1,000° and 1,100°. At this temperature part of each of the elements mentioned above is volatilised, and the higher the temperature and the longer the time during which the metal is kept melted, the smaller will be the proportion of the volatile elements retained by the gold, but they cannot be entirely removed by heat alone. This is due to the formation of compounds with gold and silver, which are not readily decomposed by heat.

The losses of gold and silver by volatilisation are small. At 1,100° their vapour pressures are insignificant, and are independent of the presence of the impurities named. If, however, a large proportion of volatile elements are present, and the mixture boils, gold and silver are carried off mechanically as spray owing to the bursting of the bubbles, and the losses may be serious. The losses are increased by the passage of a current of air over the surface of the molten metal, and consequently it is better to cover the metal with charcoal, bone ash, fluxes, or a crucible lid. The use of dust-chambers is desirable.

The losses by volatilisation or as spray are higher in some of the other refining processes, so that dust-chambers are the more necessary when these are used.

2. **Refining by Oxidation**—(*a*) *By Means of Air.*—The use of a blast of air on the surface of molten gold or silver is probably the oldest of the refining processes. The method appears to be described in the Book of Ezekiel,* to which the date B.C. 593 is assigned, and the principle is the same as in cupellation, which was described by Diodorus Siculus about B.C. 200. In cupellation, the oxides of the base metals are dissolved and slagged off by litharge, but if a blast of air is used when lead is not present in large quantities, it is necessary to add other fluxes, as many oxides are almost infusible by themselves. In 1580, Ercker recommended the metallurgist to melt brittle gold "with good Venetian borax and drive it before the bellows till it endureth the blowing."† The method is still in use at some mills to partially refine low-grade bullion, but does

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* Ezekiel, chap. xxii. 18-22.

† See Pettus on Metals (translation of Ercker's book), London, 1686, p. 218.

not seem to be employed regularly in refineries. If the oxides of iron, &c., are not slagged off, the dross collects much gold which is difficult to separate by heat alone. If large quantities of base metals are present, the slag soon covers the surface of the metal, and prevents the access of the air. It would be possible to run off the slag and keep the surface of the metal clean as in cupellation, but it seems better to pass the air through the molten metal, as in the process next described.

In certain cases, the same object may be attained by granulating the bullion, roasting the granulations spread out on trays at a red heat, with frequent stirring, and melting the product with borax and sand. After two or three repetitions of such treatment, the bullion may be refined sufficiently to be sold.*

(*b*) "*Bessemerising*."†—This is a new method, and consists in passing a stream of air or oxygen through molten bullion in clay pots by means of clay pipes similar to those used in Miller's chlorine process. The base metals are oxidised successively in the order zinc, iron, antimony, arsenic, lead, bismuth, nickel, tellurium, copper. The oxidation of these metals, however, proceeds to some extent simultaneously, some copper being oxidised before the last traces of zinc are eliminated from the bullion. The oxides rise to the surface of the metal, and are slagged off with a mixture of borax and sand. Four parts of sand and three parts of borax are enough for slagging off about six parts by weight of base metals. Lead requires less than one-third its weight of sand, zinc and copper an equal weight of sand, and iron needs 1½ times its weight of sand. The borax may be in great part replaced by about half its weight of sand, but in that case the amount of iron in the slag should be kept greater than the amount of zinc. Tin oxide may be slagged off by pearl ashes. The zinc comes off first as a sheet of flame. Then sparks, due to the formation of magnetic oxide of iron, are seen above the charge. Afterwards the action proceeds more quietly. The end of the operation is difficult to determine, except by measuring the air passed through or by dipping out part of the metal, casting it, and bending the ingot. If it is tough, only gold, silver, and copper remain unoxidised. A little silver oxidises simultaneously with copper, and if all the copper is removed into the slag, from 10 to 50 per cent. of its weight of silver is also oxidised, and passes into the slag, from which it is recovered by fusion with carbon and iron. If the operation is stopped as soon as the metal is tough, the losses of silver in the slag are small. The losses of gold in the slag are insignificant, and bullion 990 fine in silver and gold may be obtained from metal only 500 fine in an hour or two. The slag prevents loss by projection or volatilisation, and the cost is trifling. This method is used at the Mint to toughen brittle standard gold.

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* See J. S. MacArthur's remarks in a discussion at the Institution of Mining and Metallurgy, April 13, 1905.

† See paper on "Refining Gold Bullion and Cyanide Precipitates," by T. K. Rose, Institution of Mining and Metallurgy, April 13, 1905.

(*c*) *Oxidation by Nitre.*—This ancient method was described by Ercker in 1580,* who explained that the nitre must be "projected" upon the gold just before it melts, as it has little effect on molten gold. The method is still in wide use. The bullion is melted in clay crucibles, and a little nitre (potassium nitrate) or sodium nitrate is thrown on to the surface of the metal. Violent bubbling at once ensues, as heat converts nitrates into nitrites with evolution of oxygen. The nitre is pressed down with a stirrer, and the nitrites and undecomposed nitrates oxidise some of the base metals. The nitrates and oxides corrode the pots, and it is better to have a ring of bone ash next the pot, and to throw the nitre into the "eye" of metal in the centre. The oxides are absorbed by the bone ash, which protects the crucible from attack. After a minute or two, the action of the nitre moderates, and it is then, together with the bone ash, skimmed off with a ladle, and the operation repeated as often as necessary. If the metal is allowed to become pasty, so that it can be mixed with the nitre, the action is much more rapid and effective. If too much nitre is added at one time, the charge boils over, and part is lost.

Iron and zinc can be removed in this way, but the oxidation of lead is more tedious, and bismuth, tellurium, and copper are very troublesome. The losses by spirting are heavy, and large quantities of both silver and gold are entangled in the dross and skimmed off. In the treatment of cyanide precipitates the amount of these initial losses has been stated by Alfred James to be 10 per cent.,† and by J. S. MacArthur to be as high as 25 per cent.† There is little to recommend the method, which seems to remain in use from force of habit. Potassium permanganate has also been used as a substitute for nitre.‡

(*d*) *Oxidation by Metallic Oxides.*— Black oxide of copper, CuO, was formerly used in certain cases. It was mentioned by Ercker. The oxide is stirred in with the molten metal, and the whole then allowed to remain in the furnace for about half an hour, with occasional stirring. All base metals are oxidised, the cupric oxide being reduced first to cuprous oxide, and then to metallic copper. The cuprous oxide is dissolved in the metal, and so carries oxygen to all parts of the molten mass. The process is efficacious, but the gold is, of course, contaminated with the reduced copper.

The use of manganese dioxide in the Transvaal in refining gold-zinc-slimes from the cyanide process was described by

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* *Op. cit.*, p. 216.

† *Institution of Mining and Metallurgy*, April 13, 1905.

‡ W. Dowling, *Chem. Met. and Mng. Soc. of S.A.*, March, 1905.

Johnson and Caldecott in 1902.* As a cheaper alternative, the author proposed ferric oxide,† but this has apparently not been tried. In these cases, the oxides are reduced to lower oxides, but not to metals, so that the bullion is not contaminated by the products.

3. **Chlorination.**—Sal-ammoniac, NH_4Cl, is sometimes used to remove lead from gold bullion. When much lead is present, alternate additions of nitre and sal-ammoniac have been recommended. It is probable that the sal-ammoniac acts by decomposing basic compounds of lead which resist the action of nitre. Cupric chloride acts like gaseous chlorine, chloridising base metals and being reduced to cuprous chloride which is volatilised. The fineness of bullion is slightly raised by this agent, but it is not suitable for ordinary refining, although it may be used for toughening brittle standard gold or high-grade bullion containing traces of impurities. Gaseous chlorine is better. Its use is described under the heading of Miller's process of Parting, p. 407. In 1870, 40,000 ozs. of brittle standard gold were toughened at the Mint by the use of chlorine gas.‡ The charges were about 1,100 ozs. each, and the time of passage of the chlorine varied from five to seven minutes. The amount of impurities removed was from 0·1 to 0·3 per 1,000.

An old method of removing traces of impurities from brittle gold was to make repeated small additions of powdered corrosive sublimate (mercuric chloride). After each addition the door of the furnace must be at once closed, as dense poisonous fumes arise and must not be breathed by the workers. Volatile chlorides of zinc, copper, antimony, bismuth, &c., are formed and pass off, carrying with them some gold, of which there is an appreciable loss. A little corrosive sublimate sprinkled on the surface of molten gold will completely toughen every part of it without being mixed with it by stirring, even although the crucible contains several hundred ounces of the metal.

In 1866, 8,800 ozs. of brittle standard gold were toughened in this way at the Mint. It was divided into 14 charges of about 650 ozs. each, and from 12 to 16 ozs. of corrosive sublimate were added to each pot, the total amount used being 182·3 ozs. troy, or about 2 per cent. The tough bars produced weighed 8,762 ozs., the average fineness being 918·9, or 2·3 per 1,000 higher than the brittle bars. The actual loss of gold was 6·657 ozs., or 0·756 per 1,000.

4. **Sulphurisation** is said to be practised in the United States Mints;§ the following is a brief account:—It is effected in plumbago crucibles, and has for its main object the elimination

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* *Journ. Chem. and Met. Soc. of S. Africa*, July 1902, p. 21.

† *Idem*, *Oct.* 1902, p. 84.

‡ *First Annual Report of the Mint*, 1870, p. 95.

§ *Ninth Report Cal. State Min.*, 1889, p. 64.

from retorted metal of iron, when, as sometimes happens, it is present in large quantities. The metal is kept just above its melting point, the temperature being as low as possible in order to avoid unnecessary waste of sulphur by volatilisation. Sulphur is sprinkled round the edges of the molten mass, and stirred in with a graphite stirrer. If sulphur is added near the centre, particles of gold are lost by projection. Sulphide of iron is formed with great energy, and sulphide of silver also, but the latter is not produced rapidly until nearly all the iron has been already converted into sulphide. The gold is unaffected by the sulphur and subsides to the bottom. It is not usually cast by pouring, but allowed to solidify in the pot, a better separation between the gold and the matte being thus effected. The pot is turned out as soon as solidification has taken place, and the matte is broken off by a hammer, the gold being remelted and cast into a bar. The small quantity of gold taken up by the matte is separated by melting with metallic iron.

When retorted metal is infusible from the presence of large quantities of iron free from carbon, it may be refined by melting with galena, according to H. L. Sulman;* with pyrites, according to T. C. Cloud;* or with sulphur, according to W. M'Dermott.*

5. **The Use of Iron and Carbon.**—Iron is used to remove arsenic, antimony, sulphur, &c. The pot is stirred briskly with an iron rod for a few minutes. Antimonides, arsenides, or sulphides are formed and separate from the metal.

Carbon is used to assist iron to melt and to remove oxygen from bullion.

At the Philadelphia Mint it is found worth while to recover the gold from the iron tools used in stirring, dipping, &c. For this purpose they are melted down in a graphite crucible with a little charcoal to make grey-iron, and kept at a white heat for some time, after which the charge is allowed to cool slowly. Under this treatment the gold and silver separate out (an alloy containing three or four parts of silver to one part of gold being better for the purpose than pure gold), and are found at the bottom of the crucible sharply marked off from the surface of the iron, which is now quite freed from the precious metals.†

The melting under charcoal is sometimes necessary to render silver bars fit for coinage when they have been treated by nitre. When silver has been raised to a high degree of fineness, it is affected by a peculiar bubbling due to the evolution of oxygen previously absorbed from the nitre. In this case it is necessary to stir continuously with a graphite rod, keeping the surface covered with charcoal powder, until the bubbling ceases. If the metal, while still effervescing, is poured into a mould, it sprouts at the surface, and a shower of extremely minute particles are projected, often to some distance from the mould, requiring to be swept up; if the crucible is covered by a lid, very heavy effervescence ensues when the lid is lifted. In this case the silver ingots formed are not marketable, being brittle, of low density, and covered by heavy efflorescences.

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* *Institution of Mining and Metallurgy*, April 13, 1905.

† Egleston's *Metallurgy of Gold, Silver, and Mercury*, vol. ii., p. 728.

Casting the Ingots.—The operation of refining by one of the methods described above is necessary before gold can be exactly valued. When base metals other than copper are present, segregation occurs and the solidified metal is not uniform in composition. The result is that the exact value of the bullion cannot be determined, and the buyer must make some allowance to guard against loss. Nevertheless, refining is seldom attempted in extraction mills, and is often unnecessary in refineries, before parting the silver from the gold. The object at a mill is merely to melt down the bullion obtained, so as to bring it to a marketable form with as little loss as possible. With this object in view, no nitre is used, but the metal is kept covered by a layer of charcoal to prevent the formation of oxides, and the crucible is poured as soon as the charge has been fused and stirred.

When the removal of base metals has been successfully carried out the refined gold should be of a brilliant green colour, and its surface should remain quiet, without showing any iridescent films or other signs of continued oxidation.

When the metal is supposed to be tough, a small sample is dipped out and made into a thin ingot, which, after it has been cooled in water, is doubled up by hammering and its degree of toughness thus tested. It is then often remelted with copper to make up the standard alloy of the country, and again cast and hammered or cut in two with a shearing machine. The reason for doing this is that impure gold, although it may be tough when unalloyed with copper, may make brittle standard bars. If the gold is still found to be brittle, the main bulk of it left in the crucible is subjected to a repetition of its former treatment as often as is necessary, and as soon as the toughening is complete, the gold is covered with a layer of charcoal in the form of powder or lumps and thoroughly stirred before being poured.

It is necessary to stir the charge thoroughly before pouring, as the bar must be as homogeneous as possible to insure a correct assay. Since segregation may occur on cooling, assay pieces are often dipped out immediately after stirring. The subject of taking bullion assay pieces is further considered in Chapter xx. The stirring is usually done with a peculiarly shaped graphite rod, made expressly for the purpose. It is annealed carefully and raised to a full red heat before being introduced into the crucible, and is held firmly by a pair of tongs with special concave curved faces to its jaws, so as to fit the round rod. In the case of very small meltings it is sufficient to lift the crucible out of the furnace with the tongs and to give it a rotary motion just previous to pouring. In doing this, the metal must not be allowed to cool too much or the casting will be defective. It is advisable to close the dampers wholly or in part, so as to check the draught, when stirring is being done.

Meanwhile the ingot-mould in which the gold is to be cast has been prepared. It is cleaned thoroughly inside by rubbing with emery paper and oil, or with pumice stone, and wiping with an oily rag; or it may be blackleaded inside, as this prevents contact between the gold and the iron of the mould. It is then warmed by being placed on the top of the furnace; its temperature must not be sufficiently high to ignite the oil, but it should be too hot to touch with the hand. When the bullion is ready to pour, the mould is placed on a level surface, such as an iron stool, at a height above the floor of about 12 or 18 inches and a little oil poured into it. Any cheap non-volatile oil will do, whether animal or vegetable.

The crucible is then lifted from the furnace, usually with basket tongs, and when the temperature of the metal has fallen to but little above its melting point, it is poured rapidly but steadily into the mould. The crucible is then held in the inverted position for a short time, and jarred once or twice to cause the last portion of the metal to flow from it. The oil is ignited, and burns on the top of the cast metal, thus keeping it from tarnishing. In small castings, the slag is allowed to flow out and remain on the top of the metal in the mould; in large castings, the slag is usually skimmed off before pouring. Beads of metal are caught in a large iron tray with raised edges, in the centre of which the mould is placed. If the mould is clean and has been hot enough, and if enough oil has been used, a clean untarnished bar is produced. It is turned out of the mould by inversion of the latter, while still too hot to be handled, and the slag is separated by one or two light taps with a hammer. The bar is then, in many establishments, momentarily dipped into water to assist in the complete removal of the last fragments of the slag, and it is also a favourite practice to dip the bar, first into dilute sulphuric acid, and then into clean water, the bar retaining warmth enough after removal to expel all moisture. This treatment removes all tarnish, and any adherent particles of slag are then chipped off and assay pieces cut from the bar.

In some refineries large crucibles are used, and 3,000 or 4,000 ozs. of metal are refined at once. In this case, several ingot moulds are filled successively from one pot, usually by means of an iron ladle, the weight of the gold bars manufactured being usually either 200 or 400 ozs. each. Silver bars, on the other hand, are made much larger, often weighing 1,000 or 1,200 ozs.

Mixed bars of bullion (containing both gold and silver) are seldom cast of a greater weight than 600 ozs. in mills.

In conducting all these furnace operations the use of a thick pair of mittens made of sacking is to be recommended for the protection of the hands. The feet are best protected by wooden clogs.

Losses of Bullion Incurred in Melting.—The losses sustained in melting vary according to the composition of the metal. At the Royal Mint, the loss in melting without refining standard gold is about 0·2 per 1,000, partly recoverable from the "sweep." At the New York Assay Office they are found to vary from 0·5 to 1·5 per 1,000. The losses may be divided into mechanical loss, and loss by volatilisation. The mechanical loss is reduced by care in the conduct of the operation; it may be due to a number of causes. The crucible may break and its contents fall into the fire, or be scattered over the floor of the melting house when on the point of being poured. To avoid loss in this way, the ashpit may be constructed of a cast-iron tray, which can be easily scraped out. The floor of the room is made of carefully-laid flagstones, or, better still, of iron plates, in which there are no cracks or crannies capable of hiding metal beads. Projection by spirting out of the crucible may be occasioned if certain impurities, such as tellurium or antimony, are present in the bullion. Recovery of any metal lost in the ashes is effected by panning.

The slags formed in the course of refining frequently contain some small shots of metal, which may be recovered by grinding the slag finely and washing down the product in a pan or on a vanner. At the San Francisco Mint, the residue from the vannings are allowed to accumulate, and at intervals dried and fused with borax in an old graphite crucible at a high temperature. The crucible is left in the furnace over night to cool, and is then broken up, and the shots of metal at the bottom picked out. They are chiefly silver, very little gold being thus recovered. Shots of metal can be recovered with greater certainty from the slags by fusion with lead.

The crucibles, stirrers, lids, &c., also contain a certain quantity of gold and silver. After each melting they are scraped, and the scrapings panned, or, better still, calcined and fused with lead; but, in spite of this treatment, precious metals accumulate in the pots, and, when worn out, they are ground-up in a Chilian mill or other grinder, and panned, in order to separate the shots of metal. Sometimes mercury is fed into the grinding mill. The tailings from this treatment will often pay for fusion with lead, and subsequent cupellation. Certain refineries treat large quantities of such residues, with which may be included sweepings of the floor of the melting house.

Osmiridium in Gold Bars.—Gold sometimes contains osmium and iridium. As these metals remain together during the treatment, the mixture is commonly called osmiridium. If this is present in perceptible quantities, a fact which is not usually detected until after the bars have been parted, the gold is remelted in a clean crucible and kept fused at a high temperature for about half an hour, when the osmiridium will settle to the bottom of the crucible. This is due to the fact that osmiridium does not seem to form a true alloy with gold, and, being of high density and very infusible, the particles, unfused or partly fused, settle through the liquid gold. The crucible is then gently lifted out of the furnace, and the greater portion carefully but rapidly poured into a mould; the remainder, which contains almost all the osmiridium, is allowed to cool in the crucible until it has solidified, and then assayed for osmiridium. An alternative plan is to allow the whole charge to solidify in the crucible, and then to cut off the lowest portion, which is set aside. The osmiridium settles better from an alloy chiefly consisting of silver than from pure gold, and the rich bottoms are consequently melted several times with silver, the lowest part being cut off each time. The gold is thus gradually replaced by silver, which eventually forms by far the greater part of the mass. It is then granulated and parted, and the resulting powder of gold and osmiridium is treated with aqua regia, by which the gold is dissolved, and the osmiridium separated as a black powder. Such osmiridium is worth from eight to twenty shillings per ounce, and selected grains are used to make the "diamond points" of gold pens. Iridium is, however, not invariably separated from the gold bars in which it is contained, and traces can be observed in some of the refined commercial bars met with in London.

PARTING.

Parting is the separation of silver from gold. During the course of the operation the base metals are separated from both, but, as the presence of a high percentage of these base metals is injurious to the successful conduct of the processes which are chiefly in use, a preliminary refining by one of the methods already described is usually necessary. Only about 10 per cent. of base metals is permissible in the alloys when sulphuric acid is used, although a somewhat larger quantity does no harm if nitric acid is employed.

The processes of parting may be tabulated as follows:—

1. Cementation.
2. Melting with sulphide of antimony.
3. Melting with sulphur, and precipitation of the gold from the regulus by silver, iron, or litharge.
4. Parting by nitric acid.

5. Parting by sulphuric acid, sometimes called "refining."
6. A combination of these last two methods.
7. The Gutzkow process.
8. Parting by chlorine gas.
9. Parting by electrolysis.
10. Parting by aqua regia.

The first two of these methods were known to the ancients, and the third was described as early as the beginning of the 11th century.

Cementation.—In this ancient and obsolete process, gold was freed from small quantities of silver, copper, &c., contained in it. The method was mentioned by Pliny and described by Geber, who wrote in Arabic, probably in the eighth or ninth century; it is possibly still in use in some parts of the East and of South America. It consists in heating granulations of argentiferous gold mixed with a cement, consisting of two parts of brick-dust, or some similar material, and one of common salt, in pots of porous earthenware. The temperature used is a cherry-red heat, which is insufficient to melt the granulations. After about thirty-six hours' treatment, the greater part of the silver is converted into the state of chloride, and this, together with the cement, can be removed from association with the granulations by washing with water. The gold can in this way be raised to a fineness of about 850 or 900. The silver is recovered from the cement by amalgamation with mercury.*

Parting by Means of Sulphide of Antimony.—This process was also used to purify gold which contained only small quantities of silver. The alloy was repeatedly melted with sulphide of antimony, upon which the gold became alloyed with the antimony and sank to the bottom of the mass, while the silver was converted into sulphide and floated on the top, mixed with the excess of antimony sulphide added. The gold was subsequently refined by a blast of air directed upon it, the antimony being thus oxidised and volatilised. The method is now obsolete, but was in use at the Dresden Mint up to the year 1846, and gold of the fineness 993 was said to be produced in this way.

Parting by Means of Sulphur.—This method was formerly used for the purpose of concentrating the gold contained in auriferous silver in order to obtain a richer alloy. The granulated alloy was melted with sulphur and some of the silver was thus converted into a matte. The gold was then precipitated from the matte and collected in a smaller quantity of silver by fusion with pure silver, or with iron, or litharge. No attempt was made to obtain pure gold in this way, and the enriched alloy of gold and silver was parted by nitric acid. The silver was recovered from the matte by fusion with iron. The method was in use in several refineries in Europe at the beginning of the last century. The employment of sulphur in refining at the United States Mints has been already noticed, p. 384.

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* For a full account of this interesting process, as well as of the next succeeding two methods, the student is referred to Percy's *Metallurgy of Silver and Gold*, pp. 356-402.

Parting by Nitric Acid.—The first clear mention of the use of nitric acid for parting silver from gold is made by Albertus Magnus, who wrote in the thirteenth century, but the process does not appear to have been employed on a large scale until two centuries later in Venice. Here, according to an old tradition,* some Germans were employed in separating gold from Spanish silver in the fifteenth and sixteenth centuries, the art being kept secret. These refiners were not inaptly named "gold makers" by those who were unacquainted with their methods. The process was fully described by Biringuccio in his treatise,† published in 1540, and by Agricola‡ in 1556. It was first used in the Paris Mint about the year 1514, and in London at least as early as 1594, but for a long period the operations were conducted in secret in both countries, and it is supposed that this method of refining was not fully practised in England until about the middle of the eighteenth century.

Parting by means of nitric acid is conducted on the large scale in the same general manner as in the assaying of gold bullion. It consists of the following operations:—

1. Granulation of the alloys.
2. Solution of the silver in nitric acid.
3. Treatment of the gold residues, viz.:—Sweetening by washing with water, drying, melting, and casting into bars.
4. Precipitation of the silver as chloride by salt solution.
5. Reduction of the silver chloride by zinc and sulphuric acid.

Granulation of the Alloys.—The gold to be parted must be approximately free from base metals, particularly from those which are not soluble in nitric acid, such as tin, arsenic, antimony, &c. If these were present they would form insoluble oxides, which would remain with the gold, so that further refining operations would be necessary: they would, moreover, cause a great increase in the consumption of nitric acid, so, if they are present, the gold is freed from them as far as possible by melting with nitre, &c. Copper, lead, and other metals which are readily soluble in nitric acid are less obnoxious, and small percentages of these are allowed to remain, as they are not difficult to separate from the silver when in solution with it, while the presence of copper in particular is advantageous in promoting rapid dissolution of the alloy. If present in large quantities, however, even these metals would create difficulties and expense, increasing the consumption of acid.

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* Beckmann's *History of Inventions*, vol. iv., p. 578.

† *De la Pirotechnia*, Florence, 1540, Book iv., chaps. i.-v., pp. 64-71.

‡ *De re Metallica*.

The bars are melted together to form an alloy which, it was formerly believed, must contain one part of gold to three parts of silver* (hence the term "inquartation" applied to this process). In most refineries, however, the proportion of silver used is less, the minimum being $1\frac{3}{4}$ parts of silver to one of gold. At the Philadelphia Mint, the proportion is now $2\frac{1}{3}$ to 1.† Doré bars containing small quantities of gold are, of course, preferred to bars of fine silver for the purpose of alloying with the argentiferous gold bars. After the "inquarted" alloy has been thoroughly mixed by being stirred while still in the furnace, the metal is poured into copper tanks filled with cold water, which is sometimes kept cool by ice, while, in some refineries, a stream of water is kept constantly flowing through the tank. The metal is poured with a circular and wavy motion in a thin stream to prevent the formation of lumps; leafy granules and small hollow spheres are thus formed. The pouring is done either from a crucible or from a ladle, the vessel being filled from the large crucible by dipping, and held about 3 feet above the surface of the water. In the tank is a perforated copper pan, which is lifted out when the pouring is completed, and the granulations allowed to drain.

Dissolving the Granulations.—The granulated metal is heated with nitric acid in cylindrical vessels of earthenware, porcelain, or platinum. The porcelain vessels in use at the Philadelphia Mint have a capacity of about 33 gallons, and contain 2,080 ozs. of granulations: twelve such vessels are placed together on a lattice work of wood, which is laid on the bottom of lead tanks, and are surrounded by water 10 inches deep kept at the boiling temperature by means of steam. The earthenware vessels are covered by a closely-fitting lid provided with a delivery tube to carry away the fumes. Messrs. Johnson & Matthey's platinum vessels, which hold 800 ozs. of metal, are heated by separate furnaces, the fuel being coke or coal gas.

The strength of the acid used varies from that of specific gravity 1·41 to that of specific gravity 1·2. When nitric acid alone is used, about 3 lbs. of acid (sp. gr. 1·2) are used to dissolve each pound of granulations, but of this quantity the amount used in the last boiling (about 20 per cent. of the whole) is available for further use; more acid is required if there is much copper present. The first addition of acid is of about $1\frac{1}{2}$ lbs. to each pound of metal; it is kept boiling gently for about five or six hours, by which time most of the silver will have been dissolved.

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* A smaller proportion of silver, however, was used at least as early as the year 1627 in Paris. Thus Savot observes in his *Discours sur les Medalles Antiques*, Paris, 1627, chap. vii., p. 72:—"S'il n'y a beaucoup plus d'argent que d'or, l'eau n'agira aucunement: de sorte qu'il faut qu'il y ayt au moins les deux tiers d'argent, et un autre tiers d'or, et encore que l'eau soit tres-bonne: car, si elle est foible, elle n'operara point." Savot did not seem to regard this proportion of 1 to 2 as of recent introduction.

† *Report of the Director of the Mint for 1902*, p. 121.

The solution is allowed time to settle, and the hot supernatant liquid is siphoned off by a gold or glass siphon, and diluted with water to prevent the formation of crystals on cooling. The second addition consists, in some establishments, of strong acid (sp. gr. 1·414), and in others of acid of the same strength as before. The second boiling is for two or three hours only, and the third boiling for only one or two hours, the liquid being siphoned off after each boiling.

The vessels are provided with hoods and small chambers in the delivery tubes, in order to effect a partial condensation of the acid, and also to recover the small amount of silver nitrate which is carried over mechanically, owing to the violence of the disengagement of gas bubbles. The fumes are conducted to the melting furnace where they are consumed, giving up their oxygen to the fuel.

The reactions that occur are partially expressed by the following equations:—

$$\begin{aligned} 6Ag &+ 8HNO_3 &= 6AgNO_3 &+ 2NO &+ 4H_2O \\ 3Cu &+ 8HNO_3 &= 3Cu(NO_3)_2 &+ 2NO &+ 4H_2O \\ 4Zn &+ 10HNO_3 &= 4Zn(NO_3)_2 &+ N_2O &+ 5H_2O \end{aligned}$$

and similar reactions for other metals. The amount of nitrous oxide evolved increases towards the end of the operation. It is seen that silver decomposes less than its own weight of nitric acid, while copper and zinc destroy nearly three times their weight of the acid. Nitric acid of specific gravity 1·2 contains about 32 per cent. of anhydrous HNO_3, so that the quantity of acid of this strength theoretically required to dissolve 1 lb. of silver, copper, and zinc is about 2·4 lbs., 8·3 lbs., and 7·6 lbs. respectively.

Treatment of the Gold Residue.—The pulverulent gold is "sweetened" by being washed thoroughly in perforated earthenware dishes with boiling distilled water, stirring being performed with a spatula of wood, platinum, or porcelain. The gold is thus freed from nitric acid and nitrate of silver, the operation being continued until the washings show no signs of turbidity on the addition of salt. The washings are added to the first silver solutions, serving to dilute them, the dilution, as has already been observed, being necessary to prevent crystallisation on cooling. The sweetened gold is generally pressed, dried, melted, and cast into bars, which are now made of a weight of either 200 or 400 ozs. The gold thus obtained is usually of a fineness of about 997 or 998, the remainder being chiefly silver, which would not pay for extraction although part of it could be separated with a further expenditure of time, fuel, and acid. The gold is pressed by a hydraulic ram, the pressure exerted being about 800 lbs. to the square inch. The cakes of metal are dried at a cherry-red heat, and then broken up for melting.

Treatment of the Silver Solution.—The solution of nitrate of silver is diluted with water, allowed to cool, and then treated with a strong solution of salt which is regulated so as not to be in large excess, continuous agitation being kept up by revolving wooden agitators driven by steam power, or by hand paddles. When all the silver has been precipitated as chloride, the whole is allowed to settle overnight, and, in the morning, the clear solution of nitrate of soda, containing most of the base metals originally present in the alloys, is drawn off and filtered. The precipitated chloride is washed several times by decantation and agitation, and finally sweetened in wooden filters by boiling water, which incidentally dissolves out the chloride of lead. The filters are usually lined with linen or some similar material.

Reduction of the Silver Chloride.—The silver chloride is then reduced in lead-lined tanks by means of granulated zinc and water acidulated with sulphuric acid. Thirty-three pounds of commercial granulated zinc are stated to be enough to reduce 100 lbs. of silver from the chloride.* The reactions involved are as follows :—

$$\begin{array}{lllllll}
(1)\ 2AgCl & + & Zn & = & ZnCl_2 & + & 2Ag \\
(2)\ Zn & + & H_2SO_4 & = & ZnSO_4 & + & H_2 \\
(3)\ H_2 & + & 2AgCl & = & 2Ag & + & 2HCl \\
(4)\ 2HCl & + & Zn & = & ZnCl_2 & + & H_2
\end{array}$$

These reactions explain the fact that, while zinc slowly reduces silver chloride in the presence of water only, the action is quickened by the addition of free acid, by which the zinc is attacked and hydrogen evolved. Nascent hydrogen is a powerful reducing agent, and decomposes silver chloride much more rapidly than zinc does, hydrochloric acid being formed and rendered available for the production of more hydrogen. The result of this is that the action, which is at first slow, becomes more and more rapid as hydrochloric acid accumulates in the solution. The sulphuric acid is only needed to start the reaction, but, of course, the more that is added, the more quickly will the operation proceed, and if much is added, the chemical action is more violent at first than afterwards, the amount of free acid present in this case falling off. At the San Francisco Mint 1 lb. of acid of 60° B. is added for every 2 lbs. of silver to be reduced. Hydrogen is evolved copiously and is carried off by a hood and flue. Energetic stirring with wooden paddles is desirable to prevent the formation of lumps of chloride, protected by a layer of silver powder. The white chloride of silver gradually turns black as the silver is reduced. To test whether the reaction is complete, some of the silver is taken out, washed well with ammonia, filtered, and the clear solution acidified with nitric acid. A white precipitate signifies that undecomposed silver chloride is still present in the vat.

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* *Ninth Report Cal. State Min.*, 1889, p. 70.

When the reduction is complete, the vats are allowed to settle, the solution drawn off, and a little sulphuric acid added to dissolve any residue of zinc that may be present. The dark grey pulverulent silver is then washed by decantation, after which it is removed to a wooden filter, and sweetened by washing with boiling water, and finally pressed, dried, and melted into bars, which are about 998 fine. The zinc and sulphuric acid used in this process are lost, and a considerable quantity of undecomposed nitric acid is also run to waste, being contained in the solution from which the silver chloride is precipitated.

The cost of refining and parting by the nitric acid process at the United States Mints at Philadelphia and New York was formerly somewhat less than 2 cents per oz. of the parting alloy, and in San Francisco nearly 3 cents. The cost for doré silver was considerably lower. In Europe, the cost was less than in the United States. The present costs are not published.

Parting by Sulphuric Acid.—This process has now, in the majority of refineries, superseded the nitric acid method, which is much more expensive, owing to the higher cost of the acid used and of the plant required. The German chemist, Kunckel, who lived in the seventeenth century, is said to have been the first to employ sulphuric acid in parting, but it was not used on the large scale until the year 1802, when it was introduced into France by C. D'Arcet, and worked in a refinery built in Paris for the purpose. It was established in London at the Mint Refinery in 1829 by Mr. Mathison, and has been in almost continuous use there ever since, with little change, the refinery having been let on lease by the Government since 1852.

The method used varies considerably in different refineries, but essentially consists of the following operations:—

1. Mixing and granulating the alloys.
2. Dissolving the silver from the granulations by sulphuric acid.
3. Washing and melting the gold residue.
4. Precipitating the silver from its solution by means of copper.
5. Recovering the copper sulphate by crystallisation.

The account given below is a general view of the operations in various refineries, the modifications adopted not being described in most cases.

Mixing and Granulating.—The alloys must be carefully prepared so as to be of suitable composition, as otherwise difficulties are encountered. The most suitable proportion of gold in the alloy is said by Dr. Percy * to be from 18 to 25 per cent., including whatever copper there may be present; but some American writers consider the proportion of one part of gold to two and a-half parts of silver to be the most desirable, whilst at a refinery at San Francisco the alloy consists of two parts of gold to three parts of silver. This proportion was instituted when alloying silver was scarce in California and has never been abandoned, but the gold thus separated is only 990 fine, containing ten parts of silver, the maximum allowed by law in the gold coins of the United States. If the ordinary proportion of three parts of silver to one of gold is used, however, the gold can be obtained about 996 fine, and the fineness of the gold can always be increased to about 998 or 999 by fusing it, first with bisulphate of potash and subsequently with nitre. If there are only a few parts of gold per 1,000 of the alloy, it has been stated that the silver left in the gold amounts to as much as 3 or 4 per cent.; nevertheless, such an alloy, when subjected in the form of bars to the action of the acid, instead of being granulated, yields gold at San Francisco of no less than 996 fine, after one boiling only.

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* *Metallurgy of Silver and Gold*, p. 471.

The amount of base metals present in the alloy must be carefully regulated, as their sulphates are little soluble in concentrated sulphuric acid, and consequently are precipitated and interfere with the progress of the operation. Bars consisting in great part of copper are often received at the San Francisco works. These are melted with fine bars so as to reduce the proportion of copper, which must not be more than about 10 or 12 per cent. of the whole; a small amount of copper facilitates the solution of the silver. A small quantity of lead is said to assist in the solution of the copper, which is somewhat slowly attacked by concentrated sulphuric acid, and a maximum amount of 5 per cent. of lead does not interfere with the operation. From the economy with which this system of parting can be practised, silver containing only 0·5 part of gold per 1,000 can be separated from it at a profit, while the nitric acid process is unremunerative if applied to an auriferous silver alloy containing one part of gold in a thousand. At the Vienna Mint, bars are parted containing 0·9 part of gold per 1,000, and at Freiberg bars containing only 0·4 part per 1,000 are profitably treated.

In England, silver bars are passed through the parting operation, if they contain at least 2 grains of gold per troy pound, or 0·35 part per 1,000, but doré silver is not parted by itself. It is mixed with rich gold alloys.

The parting alloy is usually granulated, but at the San Francisco Refinery the doré silver is not granulated but melted and cast into bars $\frac{3}{4}$ inch thick, 9 inches wide, and 15 inches long.

Solution of the Silver.—This is usually effected in cast-iron kettles, platinum having been abandoned on account of its high cost. The iron used is fine-grained compact white iron, preferably containing 3 or 4 per cent. of phosphorus, which increases the durability, although 2 per cent. only of phosphorus is considered enough by some refiners. The kettle is slowly

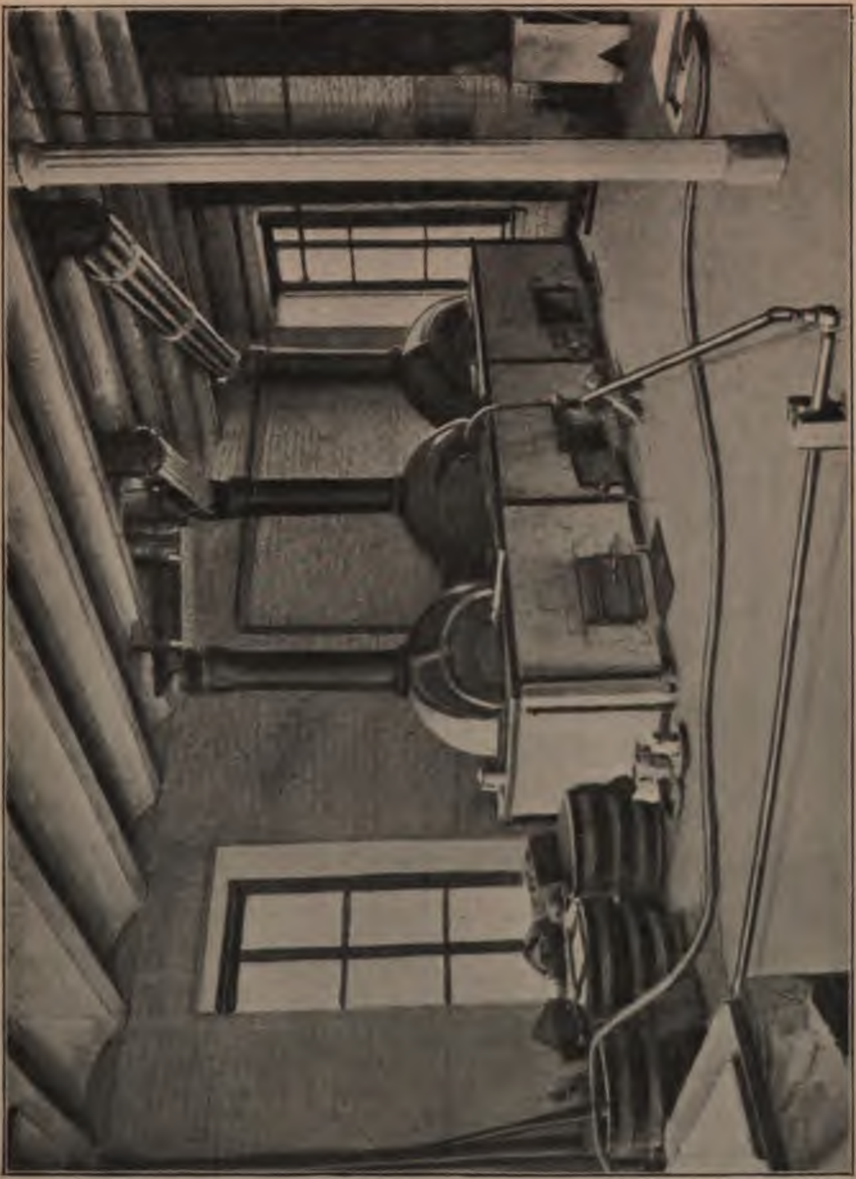

Fig. 84.—Dissolving Kettles and Hoods used in the Sulphuric Acid Parting Process. New York Assay Office.

dissolved by the acid, ferrous sulphate being formed, and, in the course of about two years, the thickness of the vessel is reduced from about 2 inches to from ¼ to ½ inch, when it is discarded. The perfect exclusion of air from the interior increases the length of life, and dilute acid must not be allowed to come in contact with the iron, as the latter is freely dissolved by it. The vessels are rectangular or cylindrical, with flat or hemispherical bottoms, the latter being preferred in Europe and the former in America. They are covered with cast-iron lids, about ½ inch in thickness, which are bolted tightly to the vessels, and have bent leaden pipes fitted to them for carrying off the fumes, which consist largely of SO_2. This is sometimes reconverted into sulphuric acid in leaden chambers arranged for the purpose. The cover has also an opening (supplied with a lid made air-tight by a water joint) through which the alloys and acids are added and the operation watched. Heat is usually supplied by a wood fire (see Fig. 84).

The charge for the pots varies from 200 to 1,000 lbs. of alloy, and the amount of acid required varies from 2 to 2½ times the weight of the alloy, depending on the composition of the latter. About one-half of the acid, which is strong commercial acid of 66° Beaumé (sp. gr. 1·85) is added at first, and the temperature cautiously raised to boiling point, when the pot is closely watched, and, if the ebullition becomes too violent, the temperature is lowered by regulating the fire and by adding cold acid a little at a time. The charge is stirred occasionally with an iron tool, particularly towards the end of the operation, when the undissolved granules of metal must be freed from the surrounding sediment, consisting of sulphates of the base metals, and exposed to the action of the acid. The ebullition gradually subsides and action ceases in about five or six hours, the presence of a greater proportion of base metals increasing the length of time required. The reactions are as follows:—

$$(1)\ 2H_2SO_4 + Ag_2 = Ag_2SO_4 + SO_2 + 2H_2O$$
$$(2)\ 2H_2SO_4 + Cu = CuSO_4 + SO_2 + 2H_2O$$

and similar reactions with tin and lead. The reactions with antimony, bismuth, zinc, and iron are more complicated. It is obvious that 63 parts of copper decompose as much sulphuric acid as 216 parts of silver. It is clear, therefore, that an increase in the percentage of copper present necessitates an increase in the amount of sulphuric acid required.

One part of sulphate of silver is soluble in ¼ part of boiling concentrated sulphuric acid, but the solubility rapidly falls off as the temperature and concentration diminish, so that 180 parts of cold acid of specific gravity 1·08 are required for the same purpose. Sulphate of copper dissolves slightly in the boiling concentrated acid, but is almost all precipitated in the form of the white anhydrous salt on cooling. Tin and zinc behave similarly, and lead makes the solution turbid and milky. The iron would not be so much attacked if it were not for the increasing dilution of the acid during the process, owing to the formation of water, which is, however, in great part boiled off as fast as it forms, or taken up by the anhydrous sulphates. The presence of copper checks the dissolution of the iron.

When the dissolution is complete, the charge is ladled into a settling pot and a few pounds of cold acid of 55° B. are added, by which the acid is cooled and diluted, and some crystals of silver sulphate are formed. These, falling to the bottom, carry down with them the suspended fine particles of gold, and so clarify the solution. If much copper is present, however, this is not necessary, as the slight cooling of the acid is enough to precipitate some sulphate of copper, which falls to the bottom and adheres to it very firmly, thus clarifying the liquid and enabling it to be poured off or ladled out very closely. The clear silver solution is then ladled out with iron ladles into lead-lined rectangular wooden vats already partly filled with hot water, in which the precipitation is subsequently effected.

Washing and Melting the Gold Residue.—The residue in the dissolving pot, if the amount of base metals present is not large, is then boiled twice more with fresh concentrated sulphuric acid added hot, after which the gold residue is hard and heavy and rapidly subsides to the bottom, and the liquors are ladled into the precipitating vat. The gold is dipped out with an iron-strainer and transferred to a lead-lined filter-box where it is thoroughly washed, first with hot dilute sulphuric acid and subsequently with boiling water, after which it is pressed, dried, and melted. It is almost always brittle, from the occurrence in it of traces of lead or tin, which are difficult to separate by sulphuric acid owing to their insolubility. These metals are eliminated by fusion with nitre or by a blast of air, and the bars thus toughened.

If the amount of base metals present is very large, the gold residues are ladled into a vessel of hot dilute sulphuric acid and boiled with it by means of steam. In this way, most of the sulphate of silver and the whole of the copper, zinc, iron, &c., remaining with the gold are rapidly dissolved. Care must be taken, however, to add the residues a little at a time, as otherwise the anhydrous sulphate of copper will form lumps, which are only slowly dissolved. The gold is then allowed to settle, and, after the solution has been drawn off, is boiled again with acid if necessary, or if it is already pure enough it is at once washed, dried, and melted.

In Europe it is not customary to attempt to obtain pure gold from auriferous silver in one operation, but the gold is concentrated in a small quantity of silver and then mixed with other alloys rich in gold and parted again. The product of gold thus obtained is purified by heating in a furnace in small iron pots with about half its weight of bisulphate of potash, by which some additional silver is converted into sulphate. The temperature is not raised much above the fusion point of the salt. The fused mass is then boiled in sulphuric acid, and again washed, dried, and melted. In the United States these methods are not used, auriferous silver being cast into slabs and parted merely by boiling with sulphuric acid; fusion with bisulphate of potash is rarely resorted to, but at the United States Mints, the residues are boiled with fresh acid five times in succession.

Precipitation of the Silver.—On pouring the sulphuric acid solution into water, most of the silver sulphate is precipitated at once in the form of small crystals, consisting of bisulphate, and the liquid must then be raised to boiling, by means of steam, in order to redissolve them. When the original alloys contain much lead this is not redissolved, and it is, therefore, necessary to let the solution settle and transfer the clear liquid to another vessel. Some particles of gold are usually found in the precipitate thus formed.

The reduction and precipitation of the silver are effected by means of copper, which takes its place in solution. The copper is usually added in the form of scrap while the liquid is being heated up by steam. The precipitation is assisted by constant stirring by means of wooden paddles. In San Francisco, however, the copper is cast into slabs, which are suspended side by side in the solution in a vertical position. The solution should be of about 24° B.; if it is much more concentrated than this, the precipitation of the silver is imperfect. The end of the reaction is detected by testing with salt solution, and when complete the stirring is stopped, the solution allowed to settle for two hours, and the clear liquid tapped into lead-lined vessels, where further settling of the suspended particles of silver takes place. The precipitate of silver is thoroughly washed with boiling water in wooden filters lined with lead, until the reaction for copper can no longer be obtained with ammonia. Care must be taken that no fragments of metallic copper remain with the silver. The metal is then pressed, dried, and melted, and is usually from 998 to 999 fine, even without fusion with nitre, when the copper plates are used for reduction.

Crystallisation of the Sulphate of Copper.—This is effected by alternate evaporation and crystallisation in lead-lined wooden tanks. The solution, which is still of 24° B., is run from the precipitating tank into the evaporating pan and concentrated to 40° B. by heating with steam; thence it is transferred to the crystallising tanks, where it is allowed to cool and remain for from ten to twelve days. The mother liquid of 36° B. is then run off and concentrated to 45° B., after which it is again allowed to crystallise, reconcentrated to 55° B., and a third crop of crystals obtained, which contain much iron. The clear acid mother liquor can now be used to dilute the solution of sulphate of silver in the dissolving pot as already described. The excess of acid in surplus liquids is neutralised with oxide of copper, more copper sulphate being thus formed.

The crystals of bluestone are found adhering to the sides and bottom of the tanks. They are detached with copper chisels, redissolved in pure water and recrystallised, the mother liquors being eventually added to the first liquor from the precipitating vats. When the liquors become over-charged with iron, the copper in them is precipitated by means of metallic iron, and they are thrown away or evaporated to get the crystals of sulphate of iron. The bluestone crystals are packed in barrels for the market. One pound of metallic copper with 1·5 lbs. of sulphuric acid of 66° B. will make 4·5 lbs. of crystallised sulphate of copper.

The cost of the process of parting by sulphuric acid in Europe is about one farthing per ounce troy of the parting alloy.

Combined Process.—At the Philadelphia Mint a combined process is used, nitric acid and sulphuric acid being employed in succession. The alloys are granulated and digested with concentrated nitric acid for six hours in the same manner as has already been described; the solution is then siphoned off, and the gold washed two or three times with distilled water, by decantation, subjected to a second boiling with strong nitric acid, and subsequently sweetened in lead-lined filters with boiling water. The gold is then introduced into cast-iron cylindrical kettles and boiled for five hours with strong sulphuric acid, the gold being stirred up with an iron rod every ten or fifteen minutes to prevent agglomeration, and the solution is then ladled out and treated as already described, p. 400. For a charge of 190 lbs. of metal, 175 lbs. of nitric acid are used in the first boiling, and 50 lbs. in the second. Some nitre is added to the sulphuric acid.

The gold is washed thoroughly and sweetened in wooden filters, boiling distilled water being poured through it until the washings will no longer redden blue litmus paper. The silver is precipitated from these washings as chloride by the addition of salt. The gold is then pressed, dried, melted, and cast into bars, which are from 998 to 999 fine.

The process is much cheaper than the nitric acid process, costing 20 per cent. less for acids, and saving some fuel. The granulations contain 100 parts of gold in 333 of the alloy. After the boiling in nitric acid very little silver is left with the gold. The cost of refining is a little over one cent per ounce.

The Gutzkow Process.—This process of parting by sulphuric acid was invented and patented by Mr. F. Gutzkow in 1867. It has been extensively worked in Germany and in San Francisco, and up to the year 1891 had been instrumental, on the authority

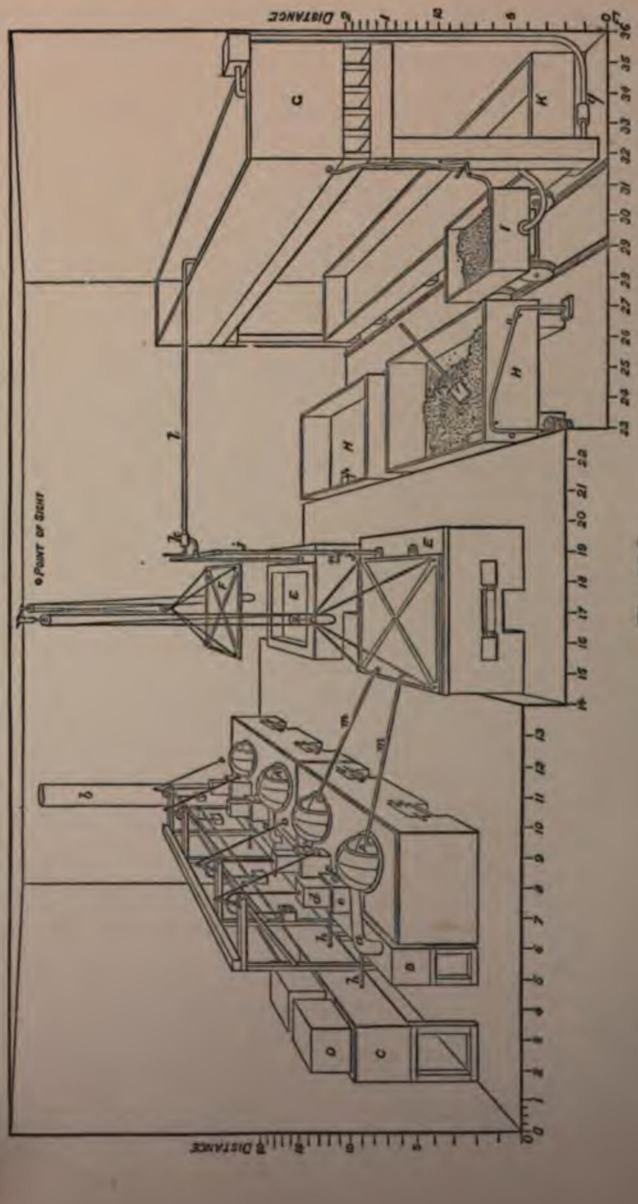
POINT OF SIGHT
DISTANCE
DISTANCE
G
K
I
H
H
E
E
F
b
m
m
C
D
B

Fig. 85.*

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* Fig. 85 is reproduced by permission of Mr. John Murray from Dr. Percy's *Metallurgy of Silver and Gold.*

of Mr. Gutzkow, in refining one hundred million dollars' worth of silver. It is fully described in Percy's *Metallurgy of Silver and Gold*, p. 479, and only a brief account will be given here. When the patent had expired, Mr. Gutzkow introduced and patented several improvements on it, which were used for some time at the Consolidated Kansas City Smelting and Refining Company's Works at Argentine, Kansas, in 1892, but subsequently abandoned, at least in part.

The original Gutzkow process, as employed at the San Francisco Assaying and Refining Works for many years, may be summarised as follows:—The bullion treated is of three kinds, viz., (1) Gold bars from retorted metal, containing about 900 parts of gold, 10 to 20 of base metals, and the remainder silver; (2) Comstock silver bars or doré bars, usually containing 20 to 100 parts of gold per 1,000; (3) base bars from the Reese River district and from pan-amalgamation of tailings, containing from 100 to 800 parts of silver, and the remainder chiefly copper, with sometimes a little gold. The gold bars (1) are alloyed with silver and granulated, but the others are cast into bars, and parted in that form. The doré bars, when prepared for solution in the acid, weigh about 100 lbs. each, and are 12 inches long, 6 inches broad, and 5 inches thick. The base ingots are melted with fine bars to reduce the average copper contents to 12 per cent., and are cast into bars 1 inch thick, the gold from which is only about 992 fine.

The boiling is done in flat-bottomed thin cast-iron kettles (A, Fig. 85), of which the bottom is only $\frac{3}{4}$ inch thick when new, and $\frac{1}{4}$ inch when worn out. The solution can be rapidly heated, owing to the thinness of the iron kettles, and 200 lbs. of alloy are dissolved in four hours by means of 300 lbs. of sulphuric acid, which comes from the tank, C, and is forced into the kettle through the pipe, *f*, by the plunger, *d*. The solution is then siphoned off through the pipe, *m*, into the tank, E, and diluted with a large quantity of hot mother liquor from a previous crystallisation, which is mainly sulphuric acid of about 58° B.; some water is also added, and the solution partially cooled, so that some crystals of silver sulphate are enabled to separate out and carry down with them the milky precipitate of lead sulphate and any suspended particles of gold; green basic sulphate of iron also settles firmly. The clear solution is then siphoned off into H and cooled to 80° F., and almost all the silver sulphate thus crystallised out. If the acid is concentrated, white soft crystals of bisulphate are formed, which is not desired; if, however, the acid is only at about 58° B., large hard yellow crystals of monosulphate, free from acid but contaminated with copper, are deposited. The mother liquor is pumped back into the tank, E, or to the original acid tank, the device employed for this purpose being to exhaust them of air, so that the acid is sucked up without passing through any valves, which would soon wear out. The crystals of sulphate of silver are transferred to the filtering box, I, by iron shovels, and a hot solution of green vitriol of 25° B. run on to them from G. This is at first mainly occupied in dissolving the sulphate of copper, and the first portion of the solution, after passing through the filtering box, is run into a storing vat, where the silver, incidentally dissolved, is precipitated by copper, and the latter subsequently recovered by means of iron. After a time, the copper being dissolved, the silver begins to be reduced, the green solution of iron turning coffee-brown; the reaction is as follows:—

$$2FeSO_4 + Ag_2SO_4 = Fe_2O_3 . 3SO_3 + 2Ag$$

The reduction may also be effected by sheets of metallic iron, which is first converted into ferrous sulphate and then into ferric sulphate, the silver being simultaneously reduced to the metallic state. The brown solution of ferric sulphate is boiled with metallic iron in K, in order to regenerate the ferrous salt. The silver is washed, pressed, dried, and melted. The gold from the original dissolving kettle is also washed in a filter, pressed, dried, and melted.

Such was the original Gutzkow process as employed in treating doré bars. Its chief advantage over the ordinary sulphuric acid process was the saving of acid. In the ordinary process, none of the acid used is saved, so that it is reduced in amount as much as possible, but does not fall below twice the weight of the silver dissolved. This reduction in the amount of acid used makes the finishing of the dissolving a difficult and delicate operation. In the Gutzkow process, however, only the acid decomposed by the silver is lost; the weight of this is about equal to that of the metal, the rest of the acid being all recovered and used over again in the boiling. Moreover, the long and tedious crystallisation of copper sulphate is avoided, and the space required for the crystallising vats saved. However, several large lead-lined vessels are required for the storage of the various solutions, and the expense of these, as well as the space required, is greatly reduced by the recent improvement described below.

*The New Gutzkow Process.**—Mr. Gutzkow has pointed out that if a large amount of acid is used for the boiling, not only is the silver more completely dissolved and the operation greatly expedited, but the presence of a high percentage of copper does not hinder the parting, as it is kept in solution by the excess of free acid. Thus, for ordinary doré silver, he proposed to use four parts of acid to one of bullion; for bars containing 20 per

* This is fully described by Mr. Gutzkow in the *Eng. and Mng. Journ.*, Feb. 28, 1891, p. 257, and May 7, 1892, p. 497, from which this account is summarised.

cent. of copper he uses six parts of acid; for still baser bullion, more acid, and so on, never losing more than one part of acid for one of bullion, and recovering the remainder.

The charge for a pot 4 feet in diameter and 3 feet in depth is 400 lbs. of doré silver: the pot is flat-bottomed, with a basin-shaped pocket or well in the centre which is useful for the collection of the gold. The bullion is first attacked by fresh acid of 66° B., run in by gravity from a large tank, and, when most of the silver has been dissolved, mother liquor from a former operation is added, a pitcher-full at a time, until the charge is completely dissolved, which takes from four to six hours. The fire is then moderated and the pot filled with mother liquor to within 1 or 2 inches of the top, when the temperature of the acid will have been so far reduced that only faint fumes are discernible. If no fumes are visible the acid is too cold and some silver sulphate will be precipitated, but otherwise the large excess of acid will keep it in solution. The well-stirred charge is now allowed to settle, which is perfectly accomplished in ten minutes, as the yellowish slowly-subsiding persulphate of iron is transformed to a greenish flocculent compound by the water in the mother liquor, and this settles quickly and carries all suspended matter to the bottom. More iron is dissolved from the kettle than in the ordinary process, owing to the greater dilution of the acid used in boiling.

The solution is now siphoned from the kettle by means of a ¾-inch gas pipe into a large cast-iron vessel, only about 1 foot deep, standing in a larger vessel which can be filled with water for cooling the charge. Steam is blown into the still hot acid solution through a lead nozzle, ⅛ inch in diameter, pointing vertically downwards. This both dilutes and warms the solution, the heating being necessary in order to prevent crystallisation of the silver consequent on the dilution. As soon as the dilution has proceeded sufficiently far to ensure the crystallisation of the hard yellow monosulphate instead of the soft white bisulphate of silver, a point which is found by dipping out small quantities at intervals, and observing their behaviour on cooling, the steam is shut off and the vat cooled with water and left all night. The silver crystals form a coating of about 1 inch thick, which is contaminated with copper sulphate if the mother liquor, by repeated use, has become saturated with it. The mother liquor is now pumped back into the acid storage tank by the creation of a vacuum, and the crystals of sulphate of silver are detached with an iron shovel and thrown into a filtering-box provided with a false bottom. Cold distilled water is sprinkled on the charge, and is allowed to filter through it and flow back into the crystallising vat, until the greater part of the free acid has been removed. The stream is then deflected into a "silver filter" where any silver is precipitated that may have been dissolved at the same time as the sulphates of iron and copper. The silver filter is a lead-lined box, partly filled with precipitated copper and provided with a false bottom. The silver separates on the top of the copper as a spongy sheet, a corresponding amount of copper being dissolved. When the crystals of silver sulphate in the first-named filtering box have been completely freed from acid, and from copper and iron sulphates, the stream of water is discontinued. The spongy sheet of silver is then removed from the "silver filter" box and treated with hot water and a few crystals of silver sulphate to dissolve the copper still retained by the sheet. During this whole operation of sweetening the crystals of silver sulphate, only about 3 per cent. of it is dissolved, as it is little soluble in cold water. The copper solution, after passing through the "silver filter" is either run to waste or precipitated by scrap iron in wooden tanks at a nearly boiling temperature.

The crystals are now dried in an iron pan which is placed above a furnace, and, after being mixed with about 5 per cent. of charcoal, they are at once charged into a hot crucible in a melting furnace. The silver sulphate is reduced at a low red heat to metallic silver, carbonic and sulphurous acid gases being evolved. By the time the temperature of melting silver is reached, these gases will have all passed away. The silver is toughened by adding nitre and borax until the so-called "boiling" indicates that the sulphur has all been eliminated, and the metal is then cast into bars.

The gold residue in the dissolving kettle contains insoluble sulphates of lead, iron, antimony, mercury, and often some copper and silver. It is ladled out and boiled with water to dissolve out the sulphates of silver, copper, iron, &c., and, after thorough filtering, it is stirred in a dish with hot water, and decanted on to a filter-cloth until the insoluble sulphates of lead, &c., have all been washed off, and the gold is left bright and clean. The gold is stored until enough is collected to make a 200-oz. bar, which is usually brittle. The material collected on the filter-cloth is re-washed once or twice to recover the particles of gold from it, and can then be reduced with charcoal and cupelled. If lead is present in the original alloy, part remains with the gold, and is dealt with in the manner which has been already described, but the greater part is carried off with the silver solution, and is deposited both while the steam is being passed in, and also subsequently during crystallisation of the sulphate of silver, which is coated with it. The sulphate of lead is removed from the crystals by stirring them well in a stream of cold water, by which the light insoluble particles of lead and antimony sulphate are carried away; it can then be collected, reduced and cupelled. Any silver that may be dissolved in the course of this washing is precipitated by copper as before.

The process is seen to differ from the original one in three essential particulars:—1. The solution is diluted with steam instead of with mother liquor, the amount of liquid in use, and consequently the number of lead-lined vats required being thus reduced. This is an important item, especially in the United States where lead-burning is expensive, owing to the existence of a powerful union. 2. The weak silver solution is precipitated at once, instead of being stored in tanks to be used again or to be precipitated at leisure. 3. The silver sulphate is reduced directly with charcoal in a crucible in the furnace. This saves the pressing of the silver and, what is of greater importance, avoids the use of the solution of sulphate of iron, which needs to be stored. The reduction in the crucible and subsequent melting requires scarcely more fuel than would be used to melt the pressed silver. One of the minor advantages of the process is said to be that no stirring is required during the boiling, owing to the large amount of acid used. This saves labour and enables the acid fumes to be more easily condensed, as they are not mixed with air, which in the ordinary way would enter through the aperture left for stirring. The exclusion of air also helps to prolong the life of the iron kettles by checking the attack on them by sulphuric acid.

Titus Ulke, however, announced* that the method of reducing sulphate of silver by heating it with charcoal, was tried by the Consolidated Kansas City Smelting and Refining Company, and abandoned, owing to the great volume of objectionable gases evolved, probably accompanied by heavy losses of silver. Gutzkow's original method of reducing sulphate of silver by a solution of proto-sulphate of iron is now used at the company's works at Argentine, and 36,000 ozs. of doré (containing on an average 4 per cent. of gold and 95 per cent. of silver) are parted daily at a mean cost (in 1895) of 0·22 cent per oz., not including cost of superintendence and office expenses.† The cost in 1892 at this works had been 0·35 cent per oz. The wages are from $2 to $2½ per day, and sulphuric acid costs one cent per pound.

Miller's Chlorine Process.—The use of chlorine gas for the purification of molten gold was first proposed by L. Thompson in 1838, and the results of his investigations were published in the *Journal of the Society of Arts* ‡ two years later. He stated that "it has long been known to chemists, that not only has gold no affinity to chlorine at red heat, but it actually parts with it at that temperature, although previously combined. . . . This, however, is not the case with those metals with which it is usually alloyed. It offers, therefore, at once an easy and certain means of separation."

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* *Mineral Industry for 1895*, p. 348.

† *Ibid.*, p. 350.

‡ Vol. liii., part i., p. 17.

In 1867, F. B. Miller, the Assayer of the Sydney Mint, applied this property of chlorine to the separation of gold from silver on the large scale, and his process has been in use at Sydney ever since, being particularly suitable for the purpose under the local conditions. Among these conditions may be mentioned the facts that acid is very costly, and that there is a scarcity of silver bullion containing small quantities of gold, while the gold produced in Australia contains but little silver. The result is that the ordinary parting processes would prove very expensive, but the chlorine process can be applied cheaply, as it requires very little acid, and is efficacious in removing small quantities of silver from gold bullion which has not been made up into alloys of definite composition. Before the introduction of the chlorine process no attempt was made to extract the silver from any of the native gold of Australia and New Zealand which was coined at the Sydney Mint. Sovereigns were manufactured containing several per cent. of silver, which replaced part of the copper used as the alloying metal. These sovereigns, some of which are still in existence, can be easily recognised by their pale tint, due to the presence of silver. Such sovereigns have not been manufactured since 1872. Besides separating the silver, the chlorine process removes the small quantities of lead, antimony, &c., which render most of the Australian retorted gold brittle, and so in one operation prepares the gold for coinage. By far the greater part of the gold produced in Australia is now deposited in the mints of Sydney, Melbourne, and Perth, and refined by this process, the amount treated in 1904 having been 3,490,930 ozs., valued at £12,820,000 out of £17,821,000 produced in Australasia.

The following description is abridged from that given by F. B. Miller,* and from later writings:—The furnace used is an ordinary melting furnace, such as has been already described. The tile cover of the furnace has a hole in the centre to allow the chlorine tubes to pass through. Clay crucibles are used, the 4-pint size being generally employed, holding about 600 or 700 ozs. of gold; they are placed inside graphite pots to prevent loss by cracking. They are glazed inside by melting borax in them to prevent them from absorbing molten chloride of silver. Graphite crucibles are said to be unsuitable, as silver chloride appears to be reduced, presumably by the hydrogen contained in them, as fast as it is formed. The crucibles are covered by loosely fitting lids, through which the clay pipe-stems of about $\frac{3}{16}$-inch bore are passed to the bottom of the crucible for the conveyance of chlorine. The pipe-stem is made red-hot before being introduced into the molten metal, as otherwise it would crack and break off. The chlorine generator consists of a stoneware jar furnished with three necks, and capable of holding from 10 to 15 gallons of liquid. The three openings are fitted with well-secured rubber plugs, through two of which two tubes are passed—viz., the safety tube, which is 8 or 10 feet high, with its open end bent over so as to deliver into a large jar, and the eduction tube, which is closed by a stopcock till it is required. The generator is partly filled with from 70 to 100 lbs. of manganese dioxide in small lumps, an amount which will suffice for many operations; hydrochloric acid is introduced through the safety tube when the gas is required. The generator is warmed by a steam jacket.

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* *Journ. Chem. Soc.*, vol. xxi. (1868), p. 506; and *Trans. Roy. Soc. of New South Wales*, Dec. 1869.

The chlorine gas is conveyed in leaden pipes to the furnaces. All joints and connections consist of well-wired india-rubber tubes, which must be protected from direct radiation from the furnace. Screw compression clamps on these rubber tubes enable the supply of chlorine to be regulated to a nicety. When the clamps are closed the gas accumulates and forces the acid up the safety tube into the vessel placed overhead, and so the further generation of gas is prevented. Two such generators and three melting furnaces are enough to refine 2,000 ozs. of gold, containing 10 per cent. of silver.

The generators being in readiness, the crucibles are slowly heated to redness, and the full charge of 600 or 700 ozs. of bullion introduced and melted, 2 or 3 ozs. of borax being sprinkled on its surface or poured on in a molten state. The chlorine is now allowed to pass slowly through the clay pipe to prevent metal from entering it, and the pipe is plunged to the bottom of the molten metal and kept there by means of a weight attached to it. The full stream of chlorine is now turned on and is heard to be bubbling into the molten metal, by which it is completely absorbed, so that no splashing and projection of the metal occurs. A height of 16 to 18 inches in the safety tube corresponds to and balances a height of 1 inch of gold in the refining crucible. The safety tube acts as an index of the pressure in the generator and of the rate of production of the gas: any leakage or the exhaustion of the acid is at once indicated by a fall of the liquid in the tube. Fresh acid is added at intervals as it is required.

When the chlorine is introduced, dense fumes at once arise from the surface of the metal owing to the formation of volatile chlorides of the base metals, which are the first to be attacked: lead gives especially dense fumes, which can be condensed on a cold object held in them. After a time these fumes cease and silver chloride is formed, very little chlorine escaping from the crucible, even if an extremely rapid current is passed into it; consequently the operation is expedited by every increase in the volume of the current. Towards the end of the operation splashing is more noticeable, and dark brownish-yellow fumes appear, consisting chiefly of free chlorine. The completion of the refining, however, is indicated by a peculiar reddish or brownish-yellow stain which is imparted to a piece of white tobacco-pipe when exposed to the action of the fumes for a moment. It is suggested by Prof. S. B. Christy that the stain contains gold. This stain appears in about one hour and a-half from the start, when 600 ozs. of gold, containing 10 per cent. of silver, are being subjected to treatment. The current of gas is then at once stopped, and the crucible lifted out of the furnace and allowed to cool sufficiently for the gold to solidify. Probably, if the operation were continued after the appearance of the brown stain, losses of gold by volatilisation would occur.

The chloride of silver, still molten, and floating on the top of the gold, is then poured off into iron moulds, and the crucible inverted on an iron table, when the red-hot cone of gold falls out. This is now fine, and after any adherent chloride of silver has been detached from it by scraping, it simply requires melting into ingots, 98 per cent. of the gold being thus at once rendered available for use. The remainder of the gold is contained in the chloride of silver, partly in the form of entangled shots of metal, but chiefly as a double chloride of silver and gold. It was formerly recovered by melting the chloride with about 10 per cent. of metallic silver, rolled to about $\frac{1}{8}$ inch in thickness. The gold is reduced by the silver and alloys with the excess settling to the bottom of the pot where it solidifies after ten minutes cooling, so that the chloride of silver can be poured off into large iron moulds, slabs suitable for reduction being thus formed.

It was found at the Sydney Mint,* that the above method of separating the gold from the silver chloride was subject to several disadvantages. In particular, although on a small scale the amount of gold in the silver could be reduced to from 0·3 to 1·0 per 1,000 by careful and continuous stirring with silver foil for a great length of time, nevertheless in practice on a large scale the results varied greatly, and the silver bullion produced usually contained from 10 to 25 parts of gold per 1,000. Several reducing agents, such as argol, resin, hydrogen, and coal gas were successively tried but were not found to give good results. Finally, the application of soda carbonate, which had been proposed by Leibius in 1868, was adopted, the method of procedure being originally as follows:—† "The argentic chloride is covered by a layer of fused borax, about $\frac{1}{4}$ inch thick, and when all is well fused, the powdered soda is sprinkled on the top of the borax, without stirring, as rapidly as the ensuing action will admit. Occasionally the top layer is dipped with a stirrer slightly underneath the molten argentic chloride, without stirring the latter. When all the necessary soda is added and the action is nearly over, the pot is covered with a lid, and left for about ten to twenty minutes to increased heat, and, when the contents are quite liquid, the pot is lifted out of the fire without previous stirring, and allowed to cool, so as to enable the argentic chloride to be poured off from the gold button at the bottom of the pot.

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* *Fourth Annual Report of the Royal Mint*, 1873. Report by A. Leibius, p. 63.

† *Loc. cit.*

"Although in several experiments all but 0·1 of gold per 1,000 was eliminated from the silver bullion produced, in no case is every trace of gold removed in one operation. To free the argentic chloride entirely from gold, producing therefore silver bullion free from gold, was, however, accomplished by subjecting the argentic chloride to a second treatment, with a small quantity of soda, in a separate boraxed clay pot, similar to the first operation.

"A convenient quantity of argentic chloride, to be treated in a No. 18 French clay pot, was found to be 230 ozs. The amount of soda required for 230 ozs. of chloride may range from 16 to 20 ozs. Less than 16 ozs. leaves too much gold in the silver, while more than 20 ozs. produces a very silvery gold button, and yet without completely freeing the argentic chloride from gold.

"The use of 18 ozs. of soda for 230 ozs. of chloride produces a gold button weighing between 30 and 35 ozs., assaying about 920 to 930, and leaves from 0·5 to 1·0 part of gold in 1,000 parts of silver bullion produced.

"With 20 ozs. of soda the results were:—Gold, about 35 ozs., assay 870-880; gold left in the silver bullion produced from 0·2 to 0·5 per 1,000.

"With 16 ozs. of soda:—Gold from 30 to 33 ozs., assay 940-950; gold left in the silver bullion, from 1·0 to 2·0 per 1,000, and sometimes as much as 6·0 per 1,000.

"To free the argentic chloride from gold, a second treatment with 3 ozs. of soda per pot of 200 ozs. chloride, containing but a minute quantity of gold, will always be found to answer, the only care required being gradual application of the soda and enough heat at the end of the operation."

The time required for the two operations is about half an hour.

"The presence of a large proportion of chloride of copper has been found to prolong the operation considerably on account of oxide of copper being formed on addition of soda, as a much greater heat is required in order to fuse the whole mass. The argentic chloride produced from base gold alloys would contain a large proportion of chloride of copper, &c., and it would be better, therefore, to reduce it direct, and dissolve the reduced metals in acid, to separate gold and silver therefrom."

The silver chloride may be assayed for gold by cupellation with lead foil and subsequent parting.

The method just described was adopted at the Sydney Mint in 1872, and at the Melbourne Mint in the following year, with excellent results.

The process of reduction of the silver chloride was devised by A. Leibius, of the Sydney Mint.* In this process, 1,400 ozs. of silver chloride are reduced in 24 hours by the apparatus, of which the following is a brief description:—Seven zinc plates, each 14 inches long, 12 inches wide, and ½ inch thick, are supported about 1¼ inches apart in a vertical position in slots in a wooden frame. Six slabs of argentic chloride, each 12 inches long, 10 inches wide, and ¾ inch thick, are suspended by loops made of silver bands, in such a way that each slab is placed between two of the zinc plates and separated from them by spaces of about ¼ inch. The silver loops are connected with silver bands on which the zinc plates rest, so that there is metallic connection between the slabs of chloride and the zinc plates. The whole is now plunged into water, to which some of the liquor from a previous operation containing chloride of zinc in solution is added as an exciting agent. Galvanic action soon begins, the liquor gets gradually warmer and a strong current is discernible. The silver chloride is gradually reduced to metallic silver, the slabs undergoing no alteration of form, and the zinc is dissolved. The slabs of silver chloride are generally free from most of the base metals, but copper, if present in the original alloy, is not volatilised in the crucible, and its chloride remains mixed with that of the silver. The two metals are now reduced together. When all action has ceased, the slabs of cupreous silver are lifted out and boiled, first in acidulated water and then in pure water, while still suspended in their silver loops. The porous metal is now ready for melting. As no acid is used the amount of zinc consumed is the theoretical quantity required by the equations—

$$2AgCl + Zn = ZnCl_2 + 2Ag$$
$$CuCl_2 + Zn = ZnCl_2 + Cu$$

The weight of zinc consumed usually amounts to from 24 to 25 per cent. of the weight of the slabs of fused chloride. The zinc plates are used over again until worn too thin for safety, after which they are melted-up and cast into new plates. They suffer no loss if the apparatus is left untouched for any length of time after the whole of the silver has been reduced.

At the Melbourne Mint in the year 1889, the zinc plates employed as described above were replaced by sheets of iron with satisfactory results. "Upon the reducing bath being heated with steam, the chloride of copper dissolving, disengages itself freely from the slabs of chloride of silver, and coming into contact with the iron is reduced, and the metallic copper falls to the bottom of the bath in large quantities, leaving the reduced silver in a much cleaner state than when zinc was used. The noxious fumes which were formerly given off on the melting of the reduced silver sponge are also avoided."*

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* *Trans. Roy. Soc. New South Wales*, 1869. The paper is given almost at full length in Percy's *Metallurgy of Silver and Gold*, p. 418.

The Chlorine Process as now practised at Melbourne.—The following description has been kindly supplied by Mr. Francis R. Power, the Assayer at the Royal Mint, Melbourne, by permission of the Deputy Master. It gives the exact methods and apparatus in use in the early part of 1896, and has been amended by Mr. Power so as to include all changes introduced up to March, 1905. As will be seen, the methods differ considerably from those described above, and from the practice at Sydney.

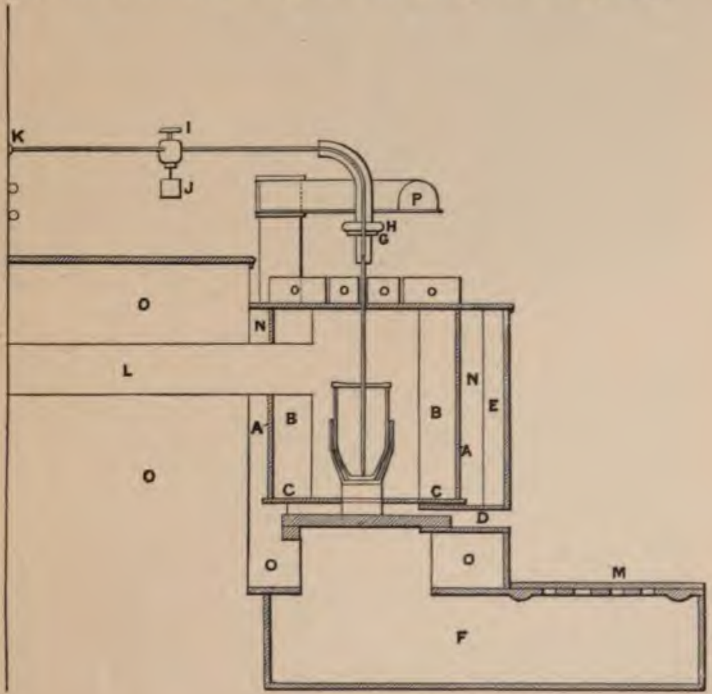

Fig. 86.

Furnaces.—These are thirteen in number. They are built cylindrically (see Fig. 86, in which one of these furnaces is shown in section, with crucible and pipe-stem in position), being more compact in this form, more easily cleaned from clinker, and more economical in fuel than the square ones. They are 12 inches in diameter and 21 inches deep. The five firebars, 1½ inches square and 18 inches long, are set in a cast-iron box,

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* *Twentieth Annual Report, Royal Mint*, 1889, p. 126.

D, 12 inches by 2 inches, which passes through the brickwork in front of the furnace, the other ends of the firebars resting on an iron bar set in the brickwork at the back of the furnace. The bars are 6 inches above the floor. The draught is obtained through a grating in the floor, which covers a portion of the ashpit, over which there slides a cast-iron plate, M, ¼ inch thick, for regulating the admission of air, and pivotted in one corner. The flue, L, is 6 inches square, and communicates with a series of five condensing chambers, 8 feet by 8 feet by 5 feet, running the length of the furnaces (42 feet), all communicating and leading to the stack, 80 feet high, common to refining and melting furnaces, which are twenty-one in all. There are three furnace covers, two of them 20 inches by 6½ inches, the third a little smaller, and all are bound with iron. The middle one is perforated by a 1-inch hole, through which the chlorine delivery pipe passes. Glenboig arched firebricks, B, 9 inches by 4½ inches, and tapering from 2¾ inches to 2 inches, are used for lining the furnaces, and are set with touching joints in an iron cylinder, A, 21½ inches in diameter, and at least ¼ to ⅜ of an inch thick, which is supported by a cast-iron plate, C, ⅝ of an inch thick, and 22 inches in diameter, with a 12-inch hole in the centre. This plate is supported by the brickwork which forms the foundation. The ashpit is a cast-iron flanged box, easily cleaned in case of an accident. Round the iron cylinder concrete, N, is rammed, the front iron plate of the furnace being shifted 2 or 3 inches in, until this is set and then moved out, thus providing an air space, E, and keeping the plates cooler. The furnace top is a plate of cast iron and, so as to facilitate repairs, should be in two pieces for each furnace, halved into one another, the hole being slightly bossed at the edge so that the firetiles may run easily on them. One piece has a hole 6 inches in diameter over which the swing ventilating hood, P, is placed by which the pot is covered when removed from the fire. This hood communicates by a passage through the brickwork with the flue. The cylindrical furnace is calculated to last for three years, the square ones lasting only eighteen months and taking three hours to reline, while the cylindrical ones take one hour.

The Crucibles, &c.—The guard pot placed for safety under the white pot and afterwards used for remelting the refined gold, is a plumbago crucible 8¼ inches high, 6 inches inside diameter, ⅝ of an inch thick at the top, and ¾ of an inch at the bottom, which is flat inside and stands on a cylindrical firebrick 5 inches in diameter and 2½ inches deep. The clay pots, fitting loosely into the guards, are 10½ inches high, 5 inches in diameter, and ⅜ of an inch thick at the top, tapering from 1 inch at the bottom. They are covered by a closely-fitting lid, dished at the top to catch any globules spirted out by too rapid a current of g

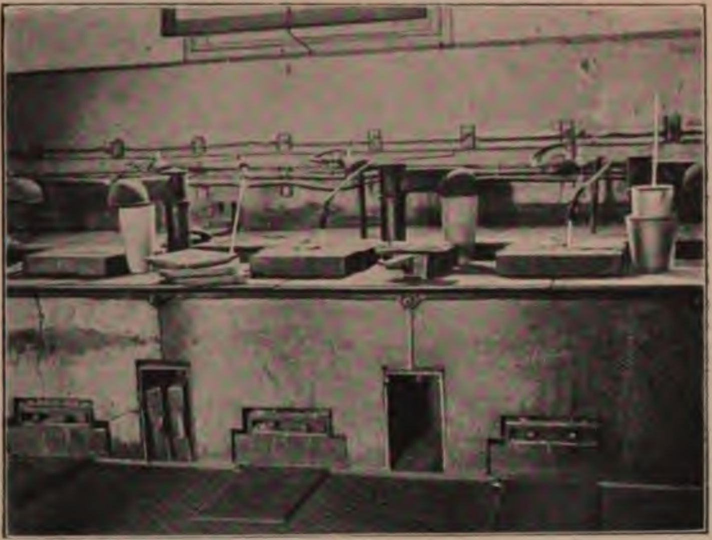

Fig. 87.

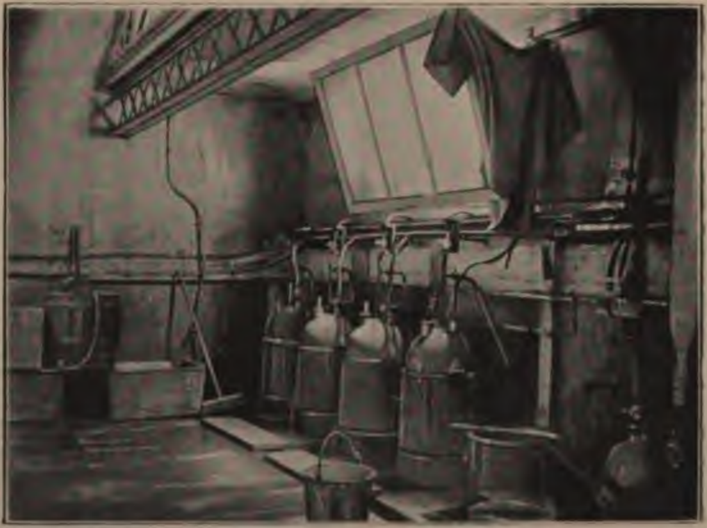

Fig. 88.

and perforated by two holes $\frac{3}{4}$ of an inch in diameter. A new pattern of lid has been introduced which has a notch in the edge for the pipe-stem to pass through, the advantage of this being the easy removal of the lid without withdrawing the pipe-stem, as is necessary with the old lids.

The pipe-stem is 24 inches long, tapering from $\frac{3}{8}$ to $\frac{1}{2}$ an inch at the end inserted into the gold, and is wedge-shaped to facilitate the escape of the chlorine when resting on the bottom of the pot. The bore of the pipe-stem is $\frac{1}{8}$ of an inch in diameter. The thin end of the pipe-stem is attached to the branch delivery pipe by a piece of $\frac{1}{2}$-inch rubber about $2\frac{1}{2}$ inches long, which connects with an ebonite junction, G, 3 inches in length, with a bore of $\frac{1}{16}$ of an inch, turned with a ring round the middle, which acts as a rest for the 8-oz. weight, H, used as a sinker for the pipe-stem. One end of the ebonite junction is $\frac{1}{2}$ an inch in diameter, the other $\frac{3}{4}$ of an inch; the latter being connected by a stout rubber tube 3 or 4 inches long to a 14-inch lead pipe ($\frac{1}{2}$ an inch in diameter) which is attached [by a rubber junction] to a glass stopcock, I, from the spigot of which a $\frac{3}{4}$-lb. lead weight, J, is suspended to prevent the pressure of gas from blowing it out. The glass stopcocks have replaced the compressor clamps, which were not satisfactory owing to the rubber cutting through, and chlorine leaking past. The rubber joints are sufficiently flexible to allow the pipe-stem to bend down into the pot or to be laid horizontally on a rest when not in use. Each furnace is provided with one glass stopcock to control the flow of gas. The cock is far enough away at the back of the furnace, to be unaffected by the heat when the firetiles are removed, and is connected by a $\frac{1}{2}$-inch lead pipe with the main pipe running along the wall. Thinner pipe-stems are found to be as serviceable as the above and do not require such careful annealing. The tubes, stopcock, &c., are somewhat diagrammatic in Fig. 86, and can be studied in Fig. 87, which is from a photograph of the furnaces by Mr. R. Law. This also shows the crucibles, guard pots, ventilating hoods, and chloride cakes.

The generators * are shown in Fig. 88, which is also from a photograph by Mr. R. Law. The pressure regulator and the reduction tank are also shown. The generators, eight in number, are three-necked cylindrical stoneware vessels with domed tops, and having a flange round the middle by which they are supported on the stoneware steam jacket 16 inches high, $16\frac{1}{2}$ inches in diameter, and $\frac{5}{8}$ of an inch thick. The domed vessel is 2 feet high. The three necks have $1\frac{3}{4}$-, $1\frac{3}{8}$-, and $1\frac{3}{8}$-inch holes, the first for charging-in the manganese ore and closed by an indiarubber plug, the second for the pipe leading to the main chlorine pipe, and the third for a branch acid supply tube, ½ an inch in diameter, fitted with a glass stopcock a foot above the neck, between which and the stopcock another ½-inch tube, the overflow, branches. The overflow tube, through which the hot waste of the generators has to pass, is provided with an ebonite stopcock which is turned off during refining. Stout combustion tubing is used. The bottom of the generator is covered by 4 inches of quartz pebbles, to prevent choking of the acid delivery pipe, which reaches to within 1 inch of the bottom of the vessel. 56 lbs. of manganese ore (about 73 per cent. peroxide), broken to ¼ to ½ inch square, is placed on the pebbles, and commercial hydrochloric acid, of specific gravity 1·16, is added as required through the acid delivery pipe by turning the glass stopcock. The acid pipe is of glass, and leads to the eight storage tanks, 20 feet above the floor, which hold 320 lbs. of acid each, and are interconnected by glass tubes luted into the bottoms; the delivery of acid, however, being from one at a time. The gas delivery pipes from the generators all connect with a 1-inch lead pipe, which leads to a distributing vessel with two necks, and partially filled with manganese chloride solution. A pressure gauge of 1 inch glass tube and 15 feet high is luted into the bottom of this vessel, and is fixed to the wall by brackets, 10 to 11 feet of the solution being required to overcome the resistance of about 7 inches of metal in the pots. The pressure in refining is equal to 5 lbs. per square inch. A four-way tube of lead or pottery is passed through the second neck of the vessel, and each arm is connected by thick rubber to glass stopcocks to which ½-inch lead pipes are joined, these pipes leading to sets of four, four and five furnaces, so that the supply of gas can be delivered to a few or all the furnaces, as desired, the subdivision being made for safety in case of a leakage or for convenience if only a few furnaces are in use. All the generators are used whether the quantity of gold to be refined be large or small, the same quantity of acid being run into each. When the flow of chlorine through the gold is stopped the acid in the generators is forced back through the overflow pipe by opening the ebonite tap. It is found necessary to have the main pipe in communication with another two-necked earthenware vessel containing such a quantity of water that when the pressure of gas exceeds the working pressure required, the end of a glass tube, passing to the bottom of the vessel and connected above the neck with an upright 4-inch lead pipe 10 feet high, becomes unsealed, and the gas escapes through the water in large bubbles, escaping through a glass pipe, inclined at an angle at the top of the lead pipe, into the air. When sufficient gas has escaped to reduce the pressure to the working limit the pipe is sealed. Thus the pipe acts automatically in keeping the pressure below such an amount as would endanger the apparatus or cause joints to leak. It is found expedient to cover all the rubber junctions in the generating room with calico and then to paint it. Protected in this way it will last until stopped up by the action of the chlorine which fills it with a lemon-yellow incrustation, at the same time reducing its thickness. All junctions are secured with copper wire where practicable.

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* These generators are now replaced by the Edwards generators described on p. 422.

Refining Operations.—The guard pot, with the clay pot in it containing 2 or 3 ozs. of fused borax, is placed in the furnace, and is heated gradually until the bottom of the clay pot is dull red. The ingots (of which the larger are slipper-shaped) to be refined, amounting in all to 650 to 720 ozs. in weight, are then placed loosely in the pot, the furnace filled with fuel, and the dampers opened. As soon as the gold is melted, which generally happens in about one and a-half hours, the boraxing of the pots being also effected at the same time, the perforated lid is put on, and the pipe-stem, previously brought carefully to a red heat to prevent cracking or flaking, is pushed to the bottom of the pot. As the pipe is being inserted, the chlorine is gently turned on to avoid stoppage of the passage through the stem by the solidification of metal in it. The supply of chlorine is controlled by the glass stopcock over the furnace, and the amount is adjusted so that the whole of the gas is absorbed and no globules of metal can be thrown up. This can usually be ascertained by feeling the pulsations of the gas through the indiarubber connections as it escapes in bubbles out of the bottom of the pipe-stem. When the gold contains much silver or base metals, the absorption of the chlorine takes place rapidly but gently, very little motion of the contents of the crucible being apparent, but when the gold to be refined is of high assay and also in all cases towards the end of refining, the gas is admitted only in a small stream, and requires careful watching to prevent spirting. When base metals are present in large quantities (over 2 per cent.) dense characteristic fumes of the chlorides of these are given off, and the metal or metals present may be generally identified by the fume or incrustation caused by the condensation of the base chlorides on the pipe or lid.

Gold containing 2 per cent. of silver and 0·5 per cent. of base metal is refined to about 995 fine in one and a-half hours, while that containing 3·5 per cent. of silver and 1·5 per cent. of base metals takes two hours. When larger percentages of silver or base metals are to be dealt with, the time taken is not proportionately longer, because, as mentioned above, a much greater stream of gas may with safety be admitted, though, in all cases, at the beginning the chlorine must be introduced gently on account of there being air in the chlorine mains, and, also, at the end of refining, the supply must be greatly reduced. When nearing completion, the "flame" issuing from the holes in the lid becomes altered in appearance, and much smaller; it now contains much chlorine mixed with small quantities of the volatile chlorides. The actual completion of the operation is generally known by the appearance of a very characteristic "flame," which is luminous, with a dark brown fringe. In case of doubt, a piece of clean pipe-stem is used as a test. It is placed, cold, for a few seconds in the issuing flame, and if the refining is finished, a clear reddish-brown stain, tending to yellow, is imparted to the test end. This stain consists of ferric oxide and chloride from the oxidation of ferrous chloride, and contains gold and sometimes chloride of silver, and is probably caused by small quantities of chloride of iron retained by the fused gold and non-volatile chlorides, from which it is freed by the unabsorbed chlorine bubbling through. Traces of copper and iron are always found in the refined gold, the bulk of the alloy being silver. As soon as the stain is found to be of the right colour, the current of gas is at once stopped, and the pipe is then withdrawn and the clay pot lifted out of the guard. If the current of gas is continued a loss of gold ensues. The pot is allowed to stand under a hood (to carry off the fumes) until the gold is set, which usually takes place in from five to seven minutes, the fact that solidification has taken place being observed by thrusting a piece of red-hot pipe-stem down through the fused chlorides. The chloride is then poured into a mould provided with a ventilating hood, which, in consequence of the high density of the fumes necessitating a sharp draught to remove them, is connected with the stack. Any borax poured off with the chlorides is allowed to remain, as it is required as a cover for the chlorides in the subsequent fusion for the separation of their gold contents. The pot is then broken, as the cone of gold will not fall out of it soon enough, and the cone of refined gold is remelted in the guard and cast into two flat ingots, 12 inches by 4 inches by 1½ inches, which, when set and still red hot, are placed on a copper lift, dipped in dilute sulphuric acid and then in water, and after removal from the water are still sufficiently hot to dry by their own heat. The broken pots are ground in a small Chilian mill and panned off, and the gold obtained is added to the "end" that is returned at the end of the day. 9,000 ozs., containing up to 10 per cent. of silver and base metals, constitute a day's refining.

An improvement has recently been introduced, by which a considerable saving of time and material is made. This is the dipping of the fused chlorides and borax from the pot while it is still in the fire, and without previous solidification of the gold, by a small clay crucible, from which they are poured into a covered mould projecting over the furnace, the drops falling back into the pot. This had previously been the practice when the percentage of silver was large, as the silver doubles its bulk on conversion to chloride, and would have overflowed. The last "dip" always contains some gold, and is poured into a separate mould, in which the metal sets at once. The chloride is thence poured into the larger mould, and the gold returned to the pot. The chloride remaining in the pot is then made into a paste with bone ash, after which the refined gold is stirred and cast into ingots, the pot being at once returned to the furnace to be used a second time.

The chlorides, which hold from 5 to 10 per cent. of gold in feathery particles, are remelted during the day in plumbago crucibles holding 300 ozs. of chloride. When fused, 7 per cent. of their weight of bicarbonate of soda is added, cautiously and without stirring, which produces a shower of globules of reduced silver, and these falling through the chlorides carry down nearly all the gold. As one addition of bicarbonate of soda does not entirely free the chlorides from gold, a second addition is made, without removing the crucible from the furnace, ten minutes being allowed after each addition. The pot is then lifted out and placed on one side to allow the metal to set, when the chlorides are poured into a mould 12 inches by 10 inches by 2 inches, practically free from gold and ready for reduction. The silvery button obtained contains from 40 to 60 per cent. of gold, and is refined on the following day.

The silvery ingot and the refined gold contain 99·95 per cent. of the gold issued in the morning for refining, the bulk of the deficiency being in the pots. The amount of gold which goes immediately into work after refining is 97·6 per cent. (an average of thirty days' refining). The amount of gold left in the silver after reduction from the chloride reaches a maximum of 1 part in 10,000, but is usually from $\frac{1}{2}$ to $\frac{1}{3}$ of this quantity.

The cakes of impure chloride, weighing about 250 ozs. each, may be colourless and translucent to brown or chocolate colour and opaque, the colour depending on the amount of copper salt present. They consist of argentic and cuprous chlorides, with traces of other chlorides and 9 per cent. of chloride of sodium from the decomposition of silver chloride by bicarbonate of soda. When cool, each cake is sewn up in a coarse flannel bag to prevent loss of any silver which may become detached during reduction, and they are then boiled with water in a wooden vat for four or five days. By exposure to air and moisture the cakes become coated with a green deposit, owing to the conversion of the cuprous chloride into cupric oxychloride and hydrated cupric chloride, and the successful removal of a large proportion of the cuprous salt in the vat is due to its solubility in a hot solution of common salt and of hydrated cupric chloride, from which it is redeposited on dilution.

The cakes for reduction are placed alternately with wrought-iron plates $\frac{1}{8}$ inch thick in a cast-iron tank lined with similar plates. The plates are prevented from touching the bags by laths of wood, otherwise the copper would be reduced in the bags, and would be difficult to separate from the silver. The reduction is slow in starting, unless either some liquor is left from a previous operation or some chloride of iron added. The bath is heated by the direct injection of steam (this is absolutely necessary), and the reduction is complete in from two to four days, though sometimes it takes longer. The time may be lessened to twenty-four hours by putting the chloride cakes in metallic connection with the iron plates. The completion of the reduction may be easily ascertained by feeling the cake, when, if any chloride be unreduced, it is left as a hard lump. The reduced silver is taken out of the bags, washed in boiling water for about an hour, and then melted with from 1 to 5 per cent. of nitre. The use of the flannel bags makes the reduced silver of high standard, as the reduced copper is thus prevented from adhering to the silver cake, from which it was found very hard to detach it without tearing off silver as well. A small percentage of reduced silver and silver chloride is found at the bottom of the tank, its presence being probably due to the solubility of the chloride in solutions of the chlorides of copper, iron, and sodium.

The mean fineness of the refined silver was only about 930 prior to the year 1889. In 1889, after the introduction of the iron plates in place of zinc, the fineness rose to [illegible]. The subsequent improvement to about [illegible] was due to the long-continued boiling in water in flannel bags. When base gold is refined the amount of copper chloride is large and is probably more easily dissolved in water, so that the resulting silver is finer than if the original bullion contained only a small amount of copper. In the latter case the copper chloride is not easily removed from the chloride cakes.

The silver contained in the gold operated on is distributed in the following manner, the mean results of the [illegible] being given:—

Silver in ingots,	[illegible] per cent.
" left in refined gold,	[illegible]
" in sweeps,	[illegible]
" unaccounted for,	[illegible]

The sweeps from the [illegible] 3 cwts. per annum, and [illegible] gold and [illegible] ozs. the silver [illegible] or volatilised [illegible]

The mean [illegible] (1891-5) was [illegible] and was [illegible] refined gold [illegible] gold in the [illegible] thousand.

The approximate cost of refining per ounce gross weight refined was as follows:—

	In 1894.	In 1895.
Material,	0·1397 of a penny.	0·1215 of a penny.
Wages,	0·1485 ,,	0·1439 ,,
	0·2882 ,,	0·2654 ,,

Half the cost for materials was for hydrochloric acid at £20 15s. per ton.

In the year 1904, 1,074,550 ozs. of rough gold were chlorinated. The average assay before refining was—

Gold,	923·0
Silver,	42·5
Base,	34·5

This was brought up to an average fineness of 994·9, costing approximately per gross ounce treated—

Stores,	0·1132 of a penny.
Wear and tear,	0·0078 ,,
Treatment of sweep,	0·0313 ,,
	0·1523 ,,
Wages, excluding supervision,	0·1222 ,,
Total,	0·2745 ,,

*Edwards' Generator.**—In 1901 a new generator, patented by T. Edwards, Pyrites Works, Ballarat, was introduced at the Melbourne Mint. The design is simple, and it has given entire satisfaction. It consists of a wrought-iron flanged cradle (see Fig. 89 which is from a working drawing kindly supplied by Mr. F. R. Power), fitted with a lead lining ⅝ inch thick, cast or hammered from sheet-lead to fit. The lining is supported at the bottom and sides by outside ribs (forming part of the casting), with openings to permit of the free circulation of the hot water used for heating the charge. The water passes between the iron case and the lining through a hole in one of the trunnions on which the generator is suspended, and escapes through a similar hole in the other trunnion. A flat lead-lined iron cover lies on the flange of the lining, and is secured by iron bolts to the flange of the cradle, a gas-tight joint being obtained by a rubber washer 3 inches wide.

There are four openings in the cover, three being 6 inches in diameter (though two of these only are absolutely necessary). These have lead flanges 1½ inches wide and lead-lined iron covers, which are kept in place by stout screws, passing through

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* Communicated by Mr. F. R. Power.

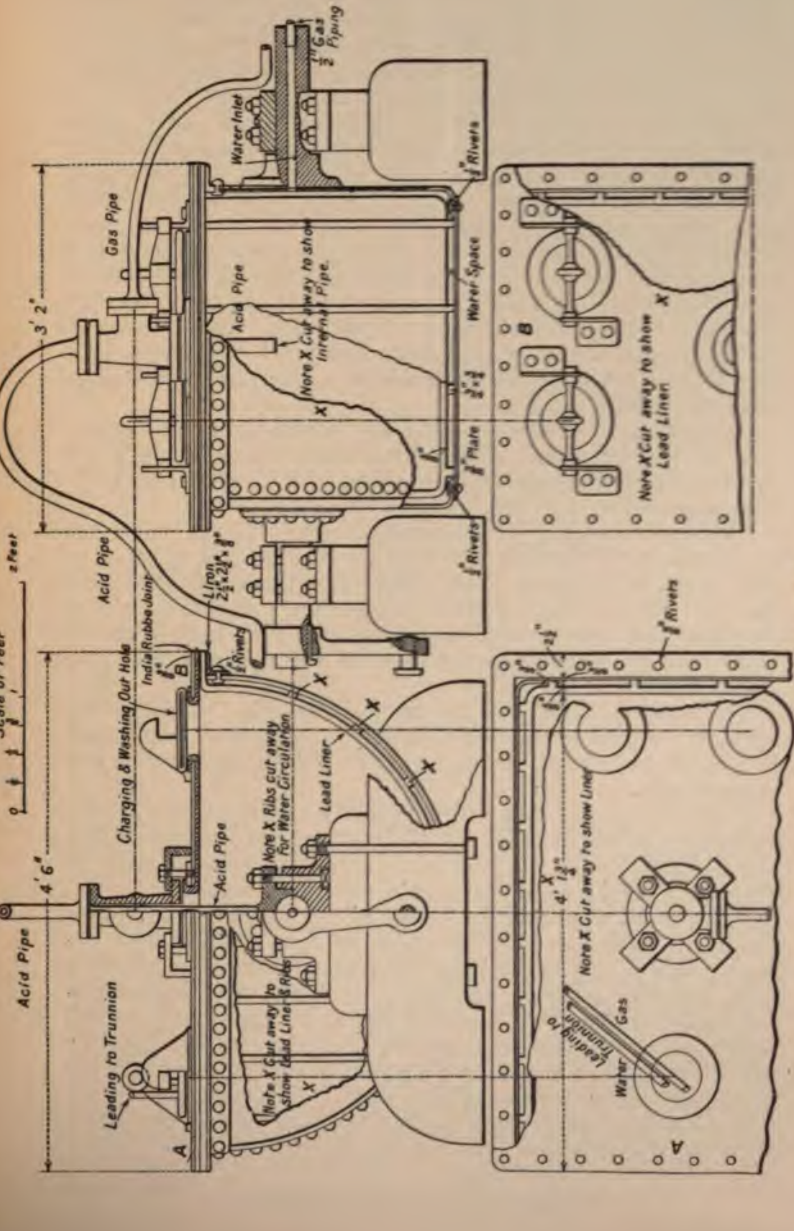
Acid Pipe
Leading to Trunnion
A
4' 6"
Scale of Feet
0 ¼ ½ ¾ 1 2 Feet
Charging & Washing Out Hole
India Rubber Joint
B
Acid Pipe
Note X Ribs cut away for Water Circulation
Lead Liner
Note X Cut away to show Lead Liner & Ribs
X
L Iron 2½" × 2½" × ⅜"
½" Rivets
3' 2"
Gas Pipe
Acid Pipe
Note X Cut away to show Internal Pipe.
Water Inlet
1½" Gas Piping
Water Space
⅜" Plate
½" Rivets
Note X Cut away to show Lead Liner
Note X Cut away to show Liner
Leading to Trunnion
Water
Gas

Fig. 89.

iron clamps bearing against iron lugs. One opening is used for charging-in and washing out, and the other has attached to its cover two pipes, one for water, which empties the generator of chlorine by forcing it through the second pipe into a second generator, when the charge of the first is exhausted.

The fourth opening, 4 inches in diameter, in the centre of the cover, has a two-way lead casting with flanged openings, fastened over it. Through this the acid is delivered by a siphon pipe, while the chlorine passes through the second opening into the pipe leading to the condensing jar, and thence to the main pipe. The acid delivery, chlorine, and water pipes are bent over to a trunnion, and fastened to it by clips, and have 2 feet of stout rubber tubing, ⅜ inch wall, inserted in their length to take up the slight twist produced when the generator rocks. The acid is fed by gravity from a graduated lead jar, 7 feet above, by a lead siphon, having an earthenware tap at the level of the generator to regulate the flow.

A crank shaft connecting with the driving arm fixed at one end of the generator gives three complete oscillations per minute, the cover making an angle of 50° with the floor. This ensures the thorough mixing of the charge.

The usual charge is 80 lbs. manganese ore (73 per cent.), 100 lbs. salt, and 10 gallons of water. The acid is allowed to flow down at intervals until about 225 lbs. of chamber acid (sp. gr. 1·83) have been added. To get the maximum yield of chlorine, the charge must be heated nearly to boiling at the end. The generator is easily emptied through a 2-inch pipe burnt into one of the covers.

The capacity is about 90 gallons, and with 100 lbs. manganese ore, 125 lbs. salt, and 265 lbs. of acid, 1,560 ozs. of base metal and silver have been removed at the Melbourne Mint in one working day.

*Preliminary Refining.**—Though the main object of the chlorine process as used at Melbourne is to produce nearly fine gold, which on alloying with copper will be tough, it is frequently employed to enable gold containing much base metal to be partially refined [so as to give a correct assay, or to enable a check assay to be made on finer metal, where the appearance of the original clips and ingot make it doubtful if a trustworthy assay is possible]. In the case of ingots containing much silver, the operation has to be carefully conducted, otherwise some silver may become converted to chloride. In partial refining it is necessary to vary the pressure of the gas according to the weight of the ingot. The pressure may be reduced automatically from the full pressure as delivered from the generator as follows (see Fig. 90):—

A is the full pressure main.

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* Communicated by Mr. F. R. Power.

B is an ebonite plug, with a pin-hole drilled through the centre, inserted in A.

C is a glass pressure gauge reaching to the bottom of a Woulff's bottle, D, which contains water, and is of about a litre capacity.

The gas passes from A through B and into D. The tap, E, allows it to pass into the melted metal, through which it bubbles as soon as the pressure in D is sufficiently great to overcome its resistance. Test pieces are dipped from time to time to determine the extent to which the refining has gone.

In a recent trial with gold-silver alloy, the silver was removed at the rate of 1 oz. per minute.

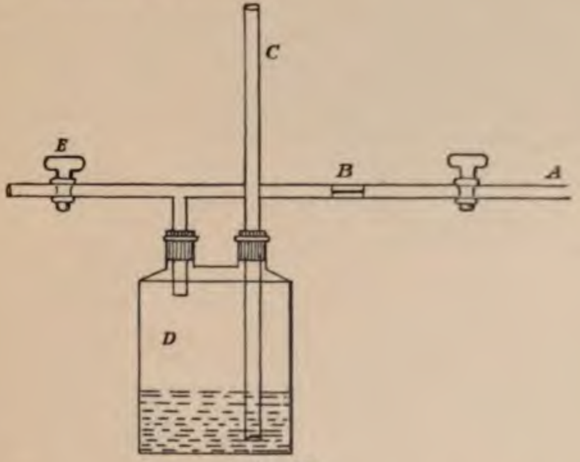

Fig. 90.

The Miller Process at the Sydney Mint.—Mr. J. M'Cutcheon, late Assayer at the Sydney Mint, wrote in 1897 that the process of freeing the chlorides from gold in use was as follows:—"The chlorides produced during the operation are separated into two classes, termed 'balers' and 'non-balers.' The first is that portion baled, or rather ladled, out during the operation, to prevent overflow; this is re-melted in quantities of 350 ozs., and whilst in the molten state, half a pound of bicarbonate of soda is projected on the surface. This has the effect of reducing some of the chloride, and the metal in sinking to the bottom of the pot carries with it all the gold. The 'non-balers,' or that portion of chloride which is poured off the refined gold when it has set, is treated as above, but 7 ozs. of granulated zinc is used instead of the bicarbonate.

"The chlorides are poured into slabs, and are now ready for the reduction process, in which the silver loops formerly used have been abandoned, iron plates being now used instead of zinc; the water is acidulated with hydrochloric acid."

The bullion treated at the Sydney Mint during 1895 contained—gold, 832·1; silver, 135·5; and base metals, 32·4 parts per 1,000.

The average fineness of the gold produced by this method at Sydney in the period of nine years from 1884 to 1892 was 995·9. The remainder is silver, which apparently cannot be profitably removed by chlorine or by any other method. The average fineness of the silver produced at Sydney was about 970 in 1893, the fineness of individual bars varying from 917 to 987. The alloying metal in the silver bars is almost all copper. The gold retained by the silver formerly amounted on an average to 1·3 parts per 1,000, but it has since been materially reduced and is now a mere trace. Analysis of the silver resulting from the refinage of gold, known originally to have contained, among the base metals in the alloy, copper, lead, antimony, arsenic, and iron, gave the following results:—

Silver,	972·3
Copper,	25·0
Gold,	2·7
Zinc and iron,	Traces.
	1,000·0

The losses of gold in the course of the process are stated to be very small, varying from 0·11 to 0·19 per 1,000; this is considerably less than would have been lost by merely toughening the gold with corrosive sublimate without parting it from the silver. The loss of silver at Sydney was about 4·25 per cent. in 1895. These losses are reduced if the amounts recovered from the flue-dust and from the ground-up crucibles are taken into account. The total cost of the process is about 1d. per oz. of crude gold at Sydney, and was about 0·65d. per oz. in Melbourne in 1873, but has since been reduced. The loss of gold by volatilisation is probably prevented from reaching the large amounts which might be expected from the results of Christy and of the author (see p. 22) by the fact that, during the whole time that the chlorine is being passed, silver and base metals are present, and, by absorbing the gas, protect the gold from its action.

Professor Thomas Price, who treated some of the Californian gold bullion by this method on a working scale in his laboratory in San Francisco, states* that, with Californian gold, which generally contains more silver than Australian gold, the gold taken up by the chloride of silver amounted to 5 per cent., and even to 10 per cent. of the total weight of the gold. For this reason, and on account of the large amount of silver bullion in the San Francisco market requiring parting, Professor Price considers that the Miller process, while technically successful with Californian gold, is hardly able to compete commercially with the ordinary sulphuric acid process. He suggests that it might be well adapted for refining the nearly pure brittle gold produced at chlorination works, where chlorine is at hand and other methods of refining are not convenient.

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* *Trans. Am. Inst. Mining Eng.*, 1888, vol. xvii., p. 30.

Parting by Electrolysis—*The Moebius Process.*—This process was patented by Bernard Moebius, in England, as long ago as the year 1884 (Dec. 16, No. 16,554), and is now in successful operation in several localities in the United States and Germany. It is said[*] to be specially suitable for refining copper bullion containing large proportions of silver and gold with small quantities of lead, platinum, and other metals, but is chiefly used in parting doré silver containing not more than 20 per 1,000 of base metals.

The apparatus required consists of a number of wooden vats coated inside with graphite paint, and filled with a solution containing 1 per cent. of nitric acid, which constitutes the electrolyte. The anodes consist of plates of bullion of about ½ inch thick, 18 inches long, and 10 inches wide, which are hung in muslin bags destined to catch the insoluble impurities after the silver, copper, &c., have been dissolved. The cathodes consist of plates of pure silver, slightly oiled to prevent adhesion of the deposited metal. These plates are continually scrubbed by a mechanical arrangement of wooden brushes by which their surfaces are kept free from loose crystals of electro-deposited silver. The loose silver falls on to trays placed below, which are removed at intervals, and the silver collected from them.

The current should have an electromotive force of from one to three volts for each vat. The copper is not deposited unless the solution becomes too weak in silver or too rich in copper, and even if some happens to be deposited, it falls into the cathode trays with the silver and does little harm, since it will be gradually redissolved if the conditions are corrected. When too much copper has accumulated in the solution the latter must be removed, the method being as follows:—The bullion anodes are replaced by carbon ones, and a weak current passed until all the silver is deposited. The silver cathodes are then replaced by copper ones, and a strong current passed so as to deposit the copper as rapidly as possible as a loose powder, which falls into a copper box. This box is connected with the cathode to prevent corrosion by the acid which is set free. The liquid thus regenerated is used again as the new electrolyte.

The process is stated to be the cheapest parting process known. If no copper were present in the bullion it is clear that there would be no consumption of acid, and it would never be necessary to change the electrolyte. Under the most favourable conditions, therefore, with water power available and the amount of copper in the bullion very small, the cost of parting doré silver would be merely nominal.

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[*] Gore's *Electrolytic Separation of Metals*, London, 1890, p. 240.

The following description of the Moebius plant which is in successful operation at the Pinos Altos Mine, Chihuahua, Mexico, is an abstract of that given by Mr. George Maynard.* The plant is capable of treating from 3,500 to 4,000 ozs. of doré silver in 24 hours, and consists of a tank 12 feet long, 2 feet wide, and 20 inches deep, divided into seven compartments. Four silver plates (the cathodes) and six doré plates (the anodes) are placed in each compartment in such a way that an anode is opposite to each face of a cathode. The trays to catch the deposited silver have perforated bottoms and are covered with asbestos cloth, and in order to facilitate cleaning-up, the electrodes, frames, anode bags, trays and brushes can all be lifted up simultaneously by a hoisting arrangement worked with a crank and handle, so that the exciting liquid alone remains in the cells.

The bullion is from 800 to 900 fine in silver, and 25 to 50 in gold, the rest being chiefly copper. The exciting liquid which at first contains 1 per cent. of nitric acid, is gradually converted into a solution of silver and copper nitrates. The silver nitrate is continually decomposed by deposition of the metal, but the copper nitrate accumulates, fresh acid being added at intervals to prevent the copper from being deposited. The silver dissolves from the anodes and is precipitated on the cathodes in the form of heavy tree-like crystals. The action of the brushes or scrapers is of vital importance to the success of the process, the advantages derived from their action being summarised as follows :—

1. The liquid is agitated and so kept homogeneous.
2. Polarisation is prevented.
3. The electrodes can be brought very near to one another, and the resistance thus reduced without any fear that short circuiting will ensue by the bridging over of the space between them by crystals of deposited silver.

The lead (as peroxide), platinum metals, antimony and other impurities remain with the gold in the bag surrounding the anodes. When the exciting liquid becomes too highly charged with copper, the solution is used instead of bluestone in the amalgamating pans of the mill, but the copper could of course be recovered as usual if it were desirable. The manual labour is all performed by the assayer and his assistant; the cathode trays are hinged, and by letting them turn on their hinges the silver is let fall into a movable tank on castors furnished with a false bottom, on which it is washed and dried, and it is then ready to be melted into bars of about 999 fine. The gold slimes are similarly washed and filtered; when they are fused, the lead is almost all slagged off.

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* *Eng. and Mng. Journ.*, May 9, 1891, p. 556.

The electric current employed is 170 amperes, the electromotive force being 8 volts; this requires an expenditure of 2½ H.P. to generate the current to refine from 3,500 to 4,000 ozs. per day. The cost of parting is said to be less than ⅓ cent per gross oz. of bullion, and the royalty is ⅓ cent per oz.

The original cost of the plant is said to have been about $6,000, including $1,000 for the silver in the cathode plates. This sum included the cost of conveying the plant to the mine, which was very high, as a journey lasting several weeks on mule-back had to be performed. The amount of silver contained at any one time in solution in the bath is about 300 ozs. The weight of the forty-two anode plates is about 4,200 ozs. when they are fresh, and this silver can be melted up and recovered about forty-eight hours after the operation has been started. The clean-up takes place once a month.

The Moebius process was also in successful operation at the works of the Pennsylvania Lead Company at Pittsburg from 1886 to 1897; here it is said that from 30,000 to 40,000 ozs. of doré bullion were refined daily. The doré bullion did not contain more than 2 per cent. of impurities (lead, copper, bismuth, &c.), and was cast into large anode plates, 1·3 cm. thick and weighing 13 to 15 kilogrammes, which are less advantageous than the light thin anodes used at Frankfort. Each anode was dissolved in about 2½ days. When the copper rose to more than 4 or 5 per cent. in the solution, part of the liquid was withdrawn, and the silver precipitated as chloride. The silver produced was from 999 to 999·5 fine, and could be raised by repeated washing to 999·85, but it is better to reduce it to 998 by adding copper, as no excess over that degree of fineness is paid for in the English market.*

At the Frankfort refinery, where this process is in use,† the general arrangements are similar to those noted above. The operation is begun with a very dilute solution of nitric acid and with a strong current of as much as 300 amperes per square metre (0·2 ampere per square inch). As the operation proceeds, copper accumulates in the liquid (the silver anodes being only about 950 fine), and reaches about 4 per cent., the silver being about 0·5 per cent., and the free nitric acid from 0·1 to 1·0 per cent. As the copper accumulates, fresh acid is added, and the current reduced to about 200 amperes per square metre. The E.M.F. required for each cell is 1·4 to 1·5 volts. The anodes are from 6 to 10 mm. thick, and each weighs about 1·5 kilogrammes; they are completely dissolved in about 36 hours, and another plate is at once substituted, the gold slimes being cleared out of the anode bag either once or twice a week. The total capacity of the Frankfort refinery (3,200 ozs. per vat in 24 hours), is about 30,000 ozs. per day, the area of the plant being only 40 feet × 45 feet.

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* *Mineral Industry for 1895*, p. 356.

† Borchers, *Electric Smelting and Refining*. Translated by W. G. M'Millan. London, 1904, p. 317.

At the Guggenheim Smelting Company, New Jersey, a continuously moving endless silver belt was formerly used as cathode, the deposited silver being brushed off by fixed brushes, so that it falls into a separate tank. The capacity is 100,000 ozs. per day, and the cost, exclusive of superintendence, office expenses, and royalty, is stated to be about ⅛ cent per oz.*

Electrolytic refining presents many advantages over acid parting, among them being the absence of noxious fumes, except during acid treatment of the gold residues, the small amount of labour and chemicals, the cleanliness and rapidity of working, and the absence of bye-products, so that the loss of metal is reduced to a minimum. In consequence of these advantages, doré silver is being more and more treated by this method.

Aqua Regia Process.†—This process was introduced at the Pretoria Mint after the Miller process had been tried and abandoned owing to the alleged difficulty of treating the gold bullion extracted by the cyanide process. In the aqua regia process the gold is dissolved and precipitated. It is made very difficult if the silver exceeds 100 parts per 1,000, and at Pretoria bullion was not treated if the silver exceeded 50 parts per 1,000. Mill gold with 80 to 110 parts of silver, and cyanide gold if it contained more than 50 parts, were melted with gold obtained by the chlorination process to reduce the silver to less than 50 parts, and granulated. Charges of 500 grammes of the granulations were then placed in each of 40 boiling flasks of 3,250 c.c. capacity, and treated with a mixture of 6 parts HCl to 1 part HNO_3. Some gold was always left undissolved to avoid loss of acid. The flasks were heated on sand baths, and at the end of the day poured into porcelain vessels holding 100 litres each.

Next morning the clear liquid was siphoned off, leaving the silver chloride at the bottom of the vessel, and transferred to a tank containing a solution of ferrous chloride. The gold thus precipitated was separated and washed, and the ferrous chloride regenerated by the addition of iron. The gold could not be toughened on melting, even by repeated additions (up to 40 times) of a few grammes of $CuCl_2$, and it was necessary to pass chlorine through it. The gold thus obtained was from 996 to 999 fine. About 300 kilogrammes of gold per month was thus refined at a cost, including subsequent coinage, of 1s. 10½d. per standard oz., 916·9 fine. The loss on the transaction was 5½d. per oz., as the Mint bought gold at £3 16s. 9½d. per standard oz., with a deduction of 4d. for refining. The floor of the refinery was slated and drained to carefully constructed gutters, and no liquids were thrown away until they were declared free from gold. The method of testing was to take 20 litres, add some acetate of lead, pass sulphuretted hydrogen, allow to settle, decant, roast the precipitate, and cupel the lead. In spite of all precautions, however, the loss of gold was heavy. This cumbrous and inconvenient process was brought to an end in 1899 by the South African War.

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* *Mineral Industry for 1895*, p. 358. *Ibid.*, 1900, p. 233.

† Begreer, *The Metallurgy of Gold on the Rand*, p. 123, Freiberg, 1898.

Wohlwill's Electrolytic Process is used at the Norddeutsche Affinerie of Hamburg, at the Deutsche Gold-und-Silber Scheideanstalt, Frankfurt, at the Royal Works at Freiberg-Halsbrücke in Saxony, and at the Philadelphia Mint. It may be described as follows:—The anode consists of impure gold, 4 mm. thick, and the electrolyte is a solution of gold chloride, containing 25 to 30 grammes of gold per litre, and from 20 to 50 c.c. of concentrated hydrochloric acid per litre, according to the strength of the current. The liquid is heated from 60° to 70° C. If the temperature is allowed to fall, or the acid is omitted, chlorine escapes. The cathode consists of pure sheet gold, and the current density is from 400 to 500 amperes per square metre. The gold, platinum, and palladium are dissolved, and the residue consists of iridium, silver chloride, and some finely-divided gold derived from the decomposition of $AuCl$. Nearly pure gold is deposited in an even coherent form on the cathode, and the solution becomes weaker in gold owing to the dissolution of impurities at the anodes, so that gold chloride must be added from time to time. The operation is carried on continuously until the amount of platinum in solution is twice that of the gold (or the palladium in solution exceeds 0·5 per cent.), when it is precipitated with ammonium chloride. The anode is stated to be reduced to one-tenth of its thickness in 24 hours, and is then melted down and recast. The electrodes are 3 cm. apart. Silver, lead, and bismuth are the cause of difficulties necessitating the brushing of the anodes and the addition of sulphuric acid.

This method offers a cheap way of preparing proof gold.

The electrolytic process is especially useful for (1) Gold mud such as that obtained from the Moebius process; (2) Argentiferous gold, if the silver and lead do not together exceed 150 parts per 1,000 and lead alone is not more than 50 parts.

The consumption of acid is about 2 per cent. of that which would be required to dissolve gold in the ordinary chemical way. There are practically no fumes, and the cost of electricity is only about three pfennige, or one penny per kilogramme of gold in Hamburg.

The platinum is accumulated until it amounts to 50 or 60 grammes per litre of solution and is then precipitated. In the year previous to May, 1900, the Hamburg refinery treated 2,000 kilogrammes of gold from all parts of the world and accumulated 1½ kilogrammes of platinum, which is worth four times the cost of refining the gold. Iridium is also completely separated from the gold. The costs at Hamburg, exclusive of superintendence and general expenses, are 1s. to 1s 6d. per kilogramme (32 ozs.) of fine gold, when working fine mud from the Moebius process, and 2s. to 2s 6d. per kilogramme of argentiferous gold. The cost of a plant for refining gold to the value of £10,000 per day would be about £1,000 in Hamburg, including £60 for a dynamo giving 150 amperes at 15 volts, and £250 for a gas engine giving 5 H.P. The space required would be about 150 square metres. A licence to use the process must be obtained from the Norddeutsche Affinerie.

The process was introduced at the Philadelphia Mint in 1902,* owing to the falling off in the amount of doré silver received. Seven cells of Berlin porcelain, each 15 inches long, 11 inches wide, and 8 inches deep, are used, filled with a gold solution containing 30 grammes of gold to the litre. In each cell are 12 anodes in multiple, each 6 inches long, 3 inches wide, and ½ inch thick, and 13 cathodes of fine gold rolled out to 0·01 inch thick. The cathodes appear to be larger than the anodes which are not completely submerged. The cells are heated from 50° to 55° on a bed of sand by means of steam pipes. The seven cells are in series with the dynamo and 100 amperes are used, the difference of potential between the terminals being from 4½ to 5 volts. One attendant is said to be enough to manage the work with the occasional help of a second. About 5,000 ozs. per week are refined with the expenditure of one horse-power.

It is evident from this account that the current density is somewhat less than 33 amperes per square foot of anode, or about 350 amperes per square metre. The gold anodes must, from their dimensions, weigh about 6,000 or 7,000 ozs. when new, so that the time of treatment is not less than a week. The cost for acid (20 cents per 1,000 ozs. of deposited gold) and power is trifling, but the labour costs are not inconsiderable, and the length of time of treatment appears to detract seriously from the value of the process.

Parting of Gold from Platinum, Iridium, &c.—The method of separating osmiridium, platiniridium, &c., from gold by remelting is described on p. 389.†

A. Baker & Co., platinum manufacturers of New Jersey, separate platinum from gold by dissolving in aqua regia, driving off acids, and precipitating the platinum with ammonium chloride.* At Freiberg, platiniferous gold is purified by melting with twice its weight of sodium bisulphate, and subsequently with a little nitre. The slags contain most of the platinum, the resulting gold being 997 to 998 fine.†

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* *Report of the Director of the U.S. Mint*, p. 121, 1902.

† *Report of the Director of the U.S. Mint*, 1885.

The best and most modern method, however, is the electrolytic one described above.

* *Mineral Industry for 1895*, p. 389.

† *Ibid.*, p. 361.

CHAPTER XIX.

THE ASSAY OF GOLD ORES.

THE assay of gold ores is almost universally conducted in the *dry* way—*i.e.*, by furnace methods. Exceptions will be noted later. The plan of operation is to concentrate the precious metal in a button of lead either (1) By fusion in a crucible; or, more rarely, (2) By scorification. The button of lead obtained by either method is then subjected to *cupellation*, by which the lead is oxidised and removed, and the resulting bead of precious metal is weighed. In these operations silver and the metals of the platinum group remain with the gold, and are subsequently separated by *inquartation* and *parting*, and, in the case of platinum and its allies, by further special methods.

The exact method to be used in any particular case varies with the composition of the ore. As a general rule, gold ores are better assayed by the fusion process, so that a comparatively large quantity of material may be operated on. Very rich ores may be assayed by scorification, the errors arising from the small amount of material used in the process being less important in their case.

The assay by scorification was preferred to pot-fusion by the German assayers of the 16th century. When fusion in a crucible was recommended, it was only as a preliminary to scorification.* Ercker describes many of the methods and precautions given in this chapter, such, for instance, as the treatment of cupriferous ores with nitric acid, the use of a salt cover in the crucible, and of "proof-Centners," which correspond to assay-tons. He also mentions the forerunner of the buck-board, and instances assay-offices in which 200 assays of ores were made in a week.

Assay by means of the Blowpipe.†—This method, though less exact than the furnace methods, is of importance, because in prospecting expeditions it is possible by its means not only to detect the gold and silver in any ore, but also to determine its amount quantitatively with fair accuracy. On such expeditions it is impossible to carry the cumbrous apparatus required to make an ordinary assay. The amount of powdered ore taken is usually 100 milligrammes, and this is mixed with borax and about 1 gramme of granulated lead. The whole is wrapped in paper and heated on charcoal in the reducing flame of a blowpipe until the fusion is complete, and then for a short time with the oxidising flame. The lead is then separated from the slag and heated on a bone-ash cupel until it is all converted into litharge. The diameter of the button of silver and gold thus obtained is carefully measured on an ivory scale, which at once gives the percentage amount in the ore. The gold is usually separated from the silver by parting in nitric acid, but Richards states [*] that the silver can be distilled off by the blowpipe, leaving a bead of pure gold, which can be measured.

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* Ercker, *Allerfurnemisten Mineralischen Eerzt u. Bergwerks Arten*, Frankfort, 1580, Book i., chap. 10; and Book ii., chap. 8. *Laws of Nature in Assaying Metals*, by Sir John Pettus, London, 1686, is a translation of Ercker's work.

† For a full description of this method, see Plattner's *Blowpipe Analysis*. Translated by Prof. Cornwall, 8th ed., New York, 1902, pp. 350-384.

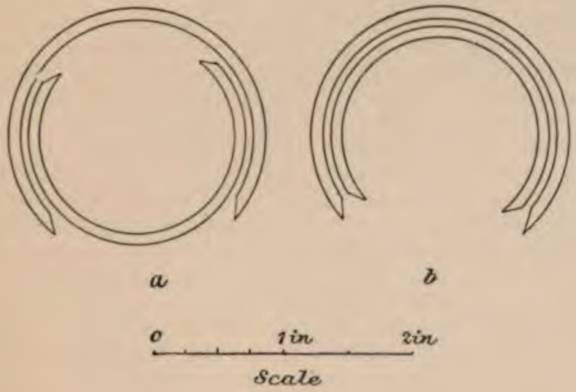

Fig. 91.

Sampling the Ore and Preparation of the Sample for Assay.—The value of an assay depends largely on the care with which a sample of the ore is selected. Sampling should be, as far as possible, automatic, and independent of the will or judgment of the assayer. The sampling of ore in place in mines is described in books on mining.[†] The taking of running samples is referred to in the section on stamp mill practice (see p. 123).

Heaps of ore, placer deposits, and impounded tailings are sometimes sampled in the same way as vat or bin charges by driving-in iron pipes[*] or cheese-taster samplers at regular intervals, and mixing the samples withdrawn in the pipes. A form of sampling iron suitable for very dry material is described by Richards.[†] It consists of two iron tubes, each with nearly half its circumference cut away. One tube fits inside the other, and is fixed to a wooden handle by a T-piece, so that it can be rotated inside the outer tube. The lower part of the outer tube is not cut, and is sharpened at the end or drawn out to a point. The tubes are driven into the ore with their openings arranged as shown in the cross-section, *a* (Fig. 91), and the inner tube is then rotated, passing through the position *b* (Fig. 91), until it is filled with ore. It is then replaced in its original position, and both tubes withdrawn. The outer tube must also be clamped to a cross-handle, to facilitate its withdrawal.

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[*] *Journal of the Franklin Institute*, June, 1896.

[†] *The Sampling and Estimation of Ore in a Mine.* T. A. Rickard. New York, 1904. See also remarks by Cordner-James, *Trans. Inst. Mng. and Met.*, vol. x. (1901), pp. 154-8; and paper by M. H. Burnham, *op. cit.*, pp. 204-218. Also Charleton, vol. ix. (1901), pp. 203-231, containing a bibliography on p. 224.

When the ore to be sampled can be moved, it is reduced in bulk either by hand or by automatic machines, and is generally crushed finer between each successive reduction of bulk.

If the ore is in course of removal by shovelling, every second, fifth, or tenth shovelful may be set aside as a sample, or the whole heap may be piled up to form a perfect cone, each shovelful being thrown as nearly as possible on the exact apex of the cone, so that the ore runs down on all sides. The heap may be made into a new cone to ensure thorough mixing, or may be at once flattened into a circular cake, and divided into four quarters, along two diameters at right angles to each other. Two opposite quarters are removed and mixed, and the operation repeated if the ore is fine enough. Coning and quartering is usually preferred to the method of setting aside particular shovelfuls (see, however, Argall, *op. cit. infra*).

Automatic samplers give better results than hand sampling, and their working costs are smaller. They are divisible into two classes—(1) those which continuously take a part of a moving stream of ore, and (2) those which momentarily take the whole stream of ore at regular intervals of time. The former class of machine is the cheaper, but is considered by Richards[‡] and Argall[§] to be useless, "because the values are never evenly distributed across the stream." In many machines a narrow scoop passes steadily across the stream of ore at regular intervals of time, and this is considered to be equivalent to the method used in the machines of class (2). The Bridgman,[‖] M‘Dermott,

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[*] *Eng. and Mng. Journ.*, vol. lxiii. (1897), p. 160. Also H. E. Nicholls, *Trans. Inst. Mng. and Met.*, vol. xiv. (1904), p. 195.

[†] *Ore Dressing*, vol. ii. (1903), p. 844.

[‡] *Ore Dressing*, vol. ii. (1903), p. 846.

[§] *Trans. Inst. of Mng. and Met.*, vol. x. (1902), p. 235.

[‖] *Trans. Am. Inst. Mng. Eng.*, vol. xx. (1891), p. 416.

Vezin,[*] Collom,[†] Brunton,[‡] Snyder,[†] Johnson,[§] and Foster-Coolidge[§] samplers are all well known.

The *Vezin sampler* is one of the few machines that give accurate results, and is stated by Argall[*] to be almost universally employed in Colorado. It consists of two sheet-steel truncated cones, *b*, *c* (Fig. 92), with their bases bolted together. The ore falling from a shoot, *a*, into a hopper, *d*, is carried outside the cone, *c*, and is delivered from the hopper through another shoot, *e*. A scoop, *f*, made of sheet-steel, with a sector-shaped opening above, is rivetted or hooked on to the cone, *b*. The angle of the sector may subtend any desired portion of the circumference of a circle, such as one-tenth (36°), or one-sixteenth (22½°). Both cones and the scoop are rotated at about 25 or 30 revolutions per minute by bevelled gearing, and when the scoop comes below the shoot, *a*, the whole stream of ore falls into the scoop, and is led into the interior of the cone, *c*, and thence to a separate truck or bucket. In this way a sample is taken about every two seconds. A Vezin sampler of about 3 feet in diameter, and requiring a fall of about 6 feet, is trifling in cost, and treats 30 or 40 tons an hour.

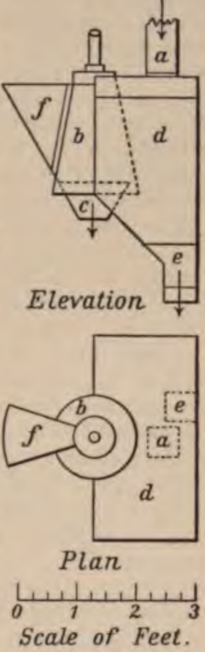

Fig. 92.

Argall recommends[||] the following course of procedure in sampling telluride ores containing 10 or 15 ozs. per ton:—From 100 short tons crushed to 1 inch cubes, take 20 tons. Crush this to ¼ inch, and take 2 tons. Crush this to 8 mesh ($= \frac{1}{16}$ inch), and reduce by "riffling" to 250 lbs. Dry and crush the sample to 30 mesh ($= \frac{1}{60}$ inch), and riffle down to 15 lbs. Crush by a sample-grinder to 90-100 mesh, and riffle down to 1 lb. This is crushed on the buck-board to pass 120 mesh ($= \frac{1}{250}$ inch), and divided for assay. Poorer and less "spotty" ores may be sampled somewhat more simply, and without such fine preliminary crushing.

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[*] *Trans. Inst. Mng. and Met.*, vol. x. (1902), pp. 240-241, with working drawings.

[†] *Ore Dressing*, pp. 846-848.

[‡] *Trans. Am. Inst. Mng. Eng.*, vol. xiii., p. 639. Hofman's *Metallurgy of Lead*, 1893, p. 55.

[§] *Mineral Industry*, vol. xi. (1903), pp. 426-430.

[||] *Op. cit.*, p. 238.

In the assay office the sample, however obtained, is further reduced in bulk by an automatic machine, similar in principle to the large samplers, such, for instance, as the Bridgman laboratory sampler,* or by the implement known as the *riffle* or sampling tin. This consists of a series of troughs arranged side by side and fastened at equal distances from each other (the width of the spaces being equal to that of the troughs) by strips of metal soldered on to their ends. An even stream of ore being let fall from a shovel on to this sampler, half is retained in the troughs, while half passes through. Careful experiments have proved that each half is representative of the whole. A repetition of the process reduces the sample to any required extent. A sampler with troughs 1 inch wide is suitable for treating materials which include lumps of not more than $\frac{1}{4}$ to $\frac{1}{3}$ inch in diameter. For finely ground materials a convenient width for the troughs is $\frac{3}{8}$ inch.

The *split shovel* is similar in principle to the riffle, and in the *Jones sampler*,† instead of troughs and spaces, two sets of troughs are substituted sloping at a high angle in opposite directions. The troughs clear themselves, discharging half the ore at each end of the machine.

When the sample has been reduced to about 20 lbs. in weight it is crushed through a 20-mesh sieve, and from 1 to 5 lbs. selected for the assay sample.‡ This sample is used for the estimation of moisture, and is then dried and finely crushed. Opinions differ as to the fineness of the sieve to be used. No one is in favour of a sieve coarser than 80-mesh, and 100-mesh is more common, but some assayers use a 200-mesh sieve. The author's experience is that all ores require crushing to 100-mesh, and that finer crushing may be dispensed with in the case of simple ores; but that complex ores, such as tellurides, should be crushed as finely as possible.

The implements employed for crushing samples of ore are various. A small *rock-breaker*, with reciprocating jaws, similar to those used on the large scale, and worked by hand or steam power, is useful for breaking down large lumps, which otherwise may be broken by a hammer. For finer pulverisation a small pair of steel-faced high-speed *rolls* may be used if steam power is available. The *sample-grinders*, resembling coffee mills, sold by makers of mining machinery, are more widely used. Such machines are adopted at large smelting or sampling works, where great numbers of samples are crushed daily. In smaller works or offices the *buck-board* (Fig. 93) is most suitable. It is a smooth plate of iron about 2 feet square with a 1-inch rim surrounding it on two or three sides. On this a *bucking hammer* is worked—a heavy piece of iron 15 to 20 lbs. in weight, with a large smooth curved face and a handle 30 inches long. It is moved about on the iron plate (on which the ore is spread) with both hands, one holding the handle, the other pressing the head downwards, the curved face being below, while an oscillatory movement is imparted by the handle. The instrument is very effective if the ore is previously broken down to the size of coarse sand in a mortar. The *pestle and mortar* are of value in breaking down samples from the size of nuts to that of coarse sand. In grinding down siliceous material, so as to enable it to pass through a fine sieve, the pestle and mortar is far inferior to the buck-board.

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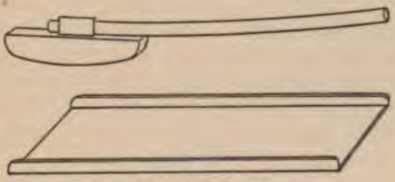
* Hofman's *Metallurgy of Lead*, 2nd ed. (1893), p. 64.

† Described by P. Argall, *Trans. Inst. Mng. and Met.*, vol. x. (1902), p. 240; and *Eng. and Mng. Journ.*, vol. lxxvi. (1903), p. 729.

‡ The views of various authorities as to weights of samples and fineness of preliminary crushings in the sampling of large lots of ore are given in Richards' *Ore Dressing*, 1st ed. (1903), p. 848.



Scale, ⅜ in. = 1 ft.

Fig. 93.

"*Metallics.*"—In many ores, both gold and auriferous silver occur native in grains or threads. These "metallics" are not readily reducible to a fine state of division, and, though a part always passes through the sieve, some of the larger pieces which have resisted abrasion fail to do so. In some assay offices part of the pulverised ore is thrown back into the mortar with the metallics, and grinding is continued until everything passes the sieve. This is a dangerous practice, as it is impossible to ensure the even distribution of metallics through the sample. The smaller pieces which pass through the sieve in the first instance constitute an unavoidable evil which is increased by every piece of metal that follows them. The safer plan is to cupel by themselves the whole of the metallics left on the sieve, and calculate their value per ton of ore independently of the result obtained by the ordinary assay. The total value of the ore is found by adding these two results together.

The prepared sample may be mixed and divided by the Bridgman divider,[*] and is stored in tin boxes or glass jars, which should be labelled by numbers, none of which are ever repeated. Before weighing out the powdered ore for assay, the whole sample should be turned out into a wide bowl or on to glazed paper, or, better still, rubber cloth (which does not crack and wear out like paper), and thoroughly mixed with a spatula. The sample should never be mixed by shaking, and care should be taken to avoid jarring it after mixing, as metallics in that case tend to settle to the bottom from their superior density, and a fair sample cannot easily be obtained. For the same reason the part of the mixed sample which is taken for assay should not be hastily shovelled on to the balance pan from the top of the pile, but a vertical slice should be taken, some of the lowest layer being carefully scraped up from the rubber cloth.

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[*] Hofman, *Metallurgy of Lead*, 2nd ed., p. 64.

FUSION OR CRUCIBLE PROCESS OF ASSAY.

This process is divided into four parts, viz.:—(1) Fusion; (2) Cupellation; (3) Parting; (4) Weighing the gold.

(1) **Fusion.**—The object of this operation is to concentrate the precious metals in a button of lead, while the whole of the remaider of the ore forms a fusible slag with suitable fluxes, in which lead sinks. The fusion is made in clay (or rarely iron) crucibles in a wind furnace, similar to, but usually smaller than, that used for melting bullion described on p. 376. Coke is generally used for fuel. Gas and petroleum furnaces are also in wide use.

In Colorado it is usual to make fusions in a muffle furnace, similar to that described below under cupellation, p. 474. The temperature required is about the same as that used in scorification. The advantages claimed are greater cleanliness and neatness and more uniformity in the conditions, the temperature of a muffle being more easily kept constant and uniform than that of an ordinary fusion furnace. Six or eight fusions can be made at one time in a large muffle.

In weighing the materials for a crucible charge the use of a set of *assay-ton* weights saves much labour in calculation. The assay-ton is a weight which contains as many milligrammes as a ton contains ounces. Thus an English or long ton of 2,240 lbs. (used in England and Australia) contains 32,666 Troy ounces, so that the corresponding assay-ton must weigh 32,666 mg. or 32·666 grammes. If the weight of the resulting bead of gold (or silver) from an assay-ton of ore is 1·5 milligrammes, then the ore contains 1·5 ozs. of gold per statute ton. If the value per short ton of 2,000 lbs. (used in North America and Africa) is required, the weight of the assay-ton is 29·166 grammes, since there are 29,166 Troy ounces in 2,000 lbs. avoirdupois. If grain weights are preferred, the weight of the assay-ton may conveniently be taken as 326·66 grains (or 291·66 grains for the short ton); the weight of the resulting bead of gold in hundredths of a grain then gives the value of the ore in ounces per ton.

Sets of weights ranging from $\frac{1}{5}$ A.T. (assay-ton) to 4 A.T. can be bought, or they can be made up from an ordinary box of decimal weights. In the following pages this system is used, one A.T. being equal to 32·666 grammes.

If grammes or grains are used as units, tables are necessary to convert the results obtained by the balance into ounces per ton. The amount of the unit used in weighing should not be lost sight of in the course of conversion. Thus if 50 grammes of ore are taken, and the resulting gold weighs 1·75 milligrammes, the unit of weight being 0·05 milligramme, the balance error corresponds to nearly 8 grains per ton, and the report should be 1 oz. 3 dwts. per ton, not 1 oz. 2 dwts. 20·8 grains as would probably be given in a table.

The weight of ore taken for assay varies according to circumstances. One A.T. is a suitable quantity if the value in gold is from 0·5 oz. to 10 ozs. or more per ton, and the balance for the final weighings is sensitive to 0·02 milligramme or less. With poor residues, 2, 4, or even 12 A.T. may be taken, and with very rich ores $\frac{1}{2}$ A.T. may suffice. If more than 3 A.T. of ore are taken, the charge is divided between two or more pots, and the resulting lead buttons scorified down and cupelled together.

The charge is made up of litharge, charcoal, and suitable fluxes, together with metallic iron. *Litharge* or red lead* is added in the proportion usually of one and a-half or two parts to two of ore. The amount of lead reduced should be from 25 to 30 grammes per A.T. of ore.

The amount of the *charcoal* added varies with the reducing power (percentage of ash, &c.) of the particular sample which is employed, as well as with the degree of oxidation of the ore. If too little charcoal is used, the amount of reduced lead will be too small, and if too much charcoal is added, the charge remains pasty and emits bubbles of combustible gas (CO). Generally from 1 to 1$\frac{1}{2}$ grammes per A.T. of ore are enough, but in some highly basic oxidised ores or roasted pyrites as much as 3 grammes of charcoal powder are required for 1 A.T. of ore, as the oxides must always be completely reduced to the lowest state of oxidation (*e.g.*, FeO) compatible with keeping them in the slag. This point is attained when nearly all the litharge

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* It is sometimes asserted that litharge (PbO) is less suitable than red lead (Pb_3O_4). The latter requires more charcoal to reduce it, and, according to Prof. Ricketts of the School of Mines, Columbia University, it is objectionable on the grounds that it oxidises silver. (*Notes on Assaying*, by Prof. Ricketts, p. 39, New York, 1897.) This drawback, however, appears to be shared by litharge. In some comparative experiments made by the author, there was no difference in results, so that it would seem that either litharge or red lead can be used with equal advantage. The amounts of silver and gold, if any, in the litharge must be determined by a separate experiment.

(about 90 per cent.) is reduced to lead. Otherwise some of the gold is oxidised and retained in the slag. If ores contain much sulphur no charcoal is used, and nitre may even be added to burn off the excess of sulphur, but in that case the pot is liable to boil over. With very large quantities of sulphides it is better to increase the quantity of soda, or, if the formation of a matte cannot be prevented in this way, to treat the matte separately, or to roast the ore before fusion. Flour and argol (acid potassium tartrate) may be used as reducing agents instead of charcoal. Their reducing power is tested beforehand.

Carbonate of soda is used to flux silica and to take up sulphur, while *borax* is valuable in basic ores to prevent corrosion of the crucible, and render the slag more liquid. The relative amounts required are judged from the appearance of the ore in the first place, and afterwards modified according to the success of the fusion. From 1 to $1\frac{1}{2}$ A.T. of sodium carbonate, and from $\frac{1}{4}$ to $\frac{1}{2}$ A.T. of borax to 1 A.T. of ore are the amounts usually required. Even when the ore is entirely composed of silica, some borax is added. The most convenient form is borax glass. Bicarbonate of soda is less convenient than the normal carbonate. If it is used the water and excess carbonic acid must be driven off by slow and cautious heating.*

Silica is used only for ores full of lime, baryta, compounds of the base metals, &c., or generally whenever the ore does not contain much quartz. It aids fusion in these cases, and protects the crucible from corrosion. From $\frac{1}{4}$ to $\frac{1}{2}$ A.T. of silica to 1 A.T. of ore is generally enough. About two parts of glass may be used instead of one part of silica. *Fluorspar* is added to the charge when the ore contains sulphates of barium or calcium, and in the fusion of cupels. Like borax, it increases the fluidity of almost any charge, but it attacks the crucible, and care must be taken to avoid deficiency of silica when it is used. In general it may be noted that for basic impurities an acid flux is used and for an acid gangue a basic flux. A cover of *common salt* is generally used to protect the charge from the variable oxidising or reducing action of the atmosphere in the furnace. Iron is added to all charges in the shape of large nails or hoop-iron, or even scrap. Sulphur, arsenic, &c., are thus kept out of the lead button. The charge for any given ore is made up by the assayer according to his judgment and experience. Much may be learnt as to the nature of the ore by the examination with a lens of unbroken lumps and, after panning, of pulverised ore. The following are some typical charges:—

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* Bettel, *Chem. and Met. Soc. of S. A.*, vol. ii., p. 467.

	Quartzose Ore.	Oxidised Ore (with 25 per cent. iron oxide).	Pyritic Ore (with 25 per cent. iron sulphide).
Ore, . . .	1 A.T.	1 A.T.	1 A.T.
Litharge, . .	1 ,,	1 ,,	1 ,,
Charcoal, . .	1·2-1·5 grms.	2·0 grms.	0·5-1·0 grm.
Sodium carbonate,	1¼ A.T.	25 ,,	1 A.T.
Borax, . .	5-10 grms.	10 ,,	10 grms.

With salt covers and hoop iron.

Methods of Operation.—The ore is weighed accurately to 0·01 gramme unless very poor when a less exact approximation is sufficient. The charcoal is always weighed with great care; the litharge is best measured by a ladle or shot measurer; the fluxes may also be measured out by a ladle more rapidly than they can be weighed. The various ingredients are thoroughly mixed together on rubber cloth or in the crucible in which the fusion is to take place. Part of the borax is often kept separate and used as a cover, being put on the top of the rest of the charge after transference to the crucible.

The crucible is carefully annealed in the ashpit of the furnace before using. It is lowered into a hollow in the fuel of the furnace made by piling the coke round an old pot and then carefully withdrawing the latter. Both ordinary and basket tongs are useful. The fire should be at a low red heat on charging-in—that is, at a temperature of 600° or 700°—and should not be urged at first. The objects to be obtained are as follows:—

(1) The gold is to be brought into such a condition that it will be readily taken up by the molten lead before the subsidence of the latter to the bottom of the pot.

(2) The mixture is to be finally brought to a state of quiet fusion, with low viscosity, so that the reduced lead may subside completely and collect into one button.

(3) There must be no mechanical losses.

Reduction of the lead begins below a red heat, and is probably almost complete before borax glass and carbonate of soda begin to melt (at about 700°) and to attack the gangue. In an ordinary charge which was heated in a muffle at 640° for twenty minutes, and then withdrawn and allowed to cool, it was found that fusion had begun though no effervescence had occurred. No lead was visible until after the charge had been panned, when a large amount of metallic lead in fine particles was recovered. The lead begins to collect into globules visible to the naked eye as soon as the charge begins to work and effervesce, but does not sink until the effervescence has proceeded for several minutes. The slower the melting the more chance is afforded of bringing the lead into intimate contact with all parts of the pasty mass and of collecting the gold from the ore. Also the slower the melting, the less chance there is that part of the charge will be projected from the pot. The final temperature need not, as a rule, be above 1,100°, and may be lower in certain cases.

The influence of the rate of heating on the result is very great. In an experiment designed to test this, four charges of a pyritic ore were made up in accordance with the table given above (p. 443). Two charges were heated very slowly, and remained in the furnace for fifty minutes before fusion was complete. The other two charges were heated as rapidly as possible, and were in the furnace for only twenty minutes. The results were as follows:—

	Heated Slowly.	Heated Rapidly.
	Ozs. per Ton.	Ozs. per Ton.
(1) Gold, . .	0·130	0·075
Silver, . .	4·42	2·33
(2) Gold, . .	0·125	0·092
Silver, . .	4·28	2·71

The mean deficiencies in the charges heated rapidly were thus:—Gold, 34 per cent.; silver, 42 per cent., but the results were also less regular than when slow heating was used.

The effervescence is due chiefly to the escape of carbonic acid from the sodium carbonate as it unites with silica, and is of value in moving about the lead globules.

After a lapse of from 40 to 50 minutes the charge is in a state of tranquil fusion, with the exception perhaps of slight action round the sides or next the iron, if any is used. The crucible is then lifted out of the fire by the tongs, the nails withdrawn and any lead adhering to them shaken off into the pot. The pot may now be tapped on the floor to assist the lead to settle in the slag, allowed to cool and broken by a hammer to extract the lead button, or the charge may be at once poured into an iron mould. The mould must be cleaned, blackleaded, and warmed before being used, and, after pouring, it is tapped on the bench to collect the lead at the bottom.

If the charge does not fuse completely, so that the slag is pasty or has lumps in it, it is advisable to recommence the assay, making such alterations in the charge as experience suggests, having regard to the considerations mentioned above, p. 442.

When the mould is quite cold its contents are readily separated from it if the precautions mentioned above are taken, and the slag is detached from the lead button with a hammer on an anvil. The slag should be glassy and homogeneous; if it is streaked it is probable that the fusion has not been perfect. It is green and transparent if the ore is nearly pure quartz; black and opaque if much iron is present; but is red if the ore contains much copper.

The lead button should be soft and malleable. If it is hard or brittle it may contain sulphur, arsenic, antimony, copper, &c. Sulphur and arsenic are kept from entering the lead by the addition of iron, and may form with the iron a regulus or speiss, which separates as hard blackish-grey or white layers found just above the lead. They are richer than the slag, and may often yield appreciable quantities of gold on further treatment by scorification or roasting and fusion. If the quantity of sodium carbonate is large enough, the sulphur is retained in the slag in the form of sodium and ferrous sulphides. Antimony makes the lead hard, white, brittle and sonorous; according to Rivot, it is not a source of loss if forming less than 1 per cent. of the lead button (see under Cupellation below). The lead must be completely freed from the slag, very small quantities of the latter interfering seriously with the cupellation, forming a scoria and occasioning loss.

Roasting before Fusion.—Ores containing large quantities of sulphur, arsenic, or antimony may sometimes be roasted with advantage as a preliminary to fusion. Roasting is effected in shallow circular clay dishes, in a muffle, or in the crucibles in which the fusion is afterwards performed. The temperature must be kept low at first and the ore frequently stirred with an iron wire or spatula, to prevent fritting, and to expose fresh surfaces to the air. The roasting takes place in two stages: at first, sulphur dioxide, arsenious oxide (As_2O_3), and antimonious oxide (Sb_2O_3) are formed and volatilised, the sulphur burning with a blue flame. The formation of lumps is most to be feared during the first few minutes of the operation, and can scarcely be prevented if much sulphide of antimony is present; in this case an equal weight or more of pure silver sand is mixed with the crushed ore before charging it into the muffle.

After a time the blue flame disappears, the odour becomes less strong, and sulphates, arseniates, and antimoniates form. By raising the temperature sulphates are decomposed, but arseniates and antimoniates are stable at high temperatures and cause loss of silver in the fusion. To prevent their formation the ore should be roasted in a coke furnace, starting to heat it very gradually and admitting a limited supply of air. In all cases the roasting is nearly complete when the glow caused by stirring is shown only by a few specks of ore; the temperature may then be raised to a strong red heat without danger of fusion. The operation is complete when the ore remains of a uniform colour on stirring. Arseniates and antimoniates may be in part removed by re-roasting with powdered charcoal in a covered dish. The fusion of roasted ores requires more charcoal powder than raw ores, the amount needed being sometimes as much as 3 grammes per A.T. of ore. In general, roasting is to be deprecated, owing to the unavoidable loss by dusting. The results are usually lower than those obtained by other methods, and the operation requires the expenditure of much time and attention. It is useful as a second method in cases of special importance.

Cleaning the Slag.—The slags of very rich ores may retain enough gold to necessitate further treatment. The slag is roughly crushed and fused with 1 A.T. of litharge, 15 to 20 grains of charcoal and a little carbonate of soda, the same crucible being used again, or the slag may be fused and a mixture of litharge and charcoal thrown on to its surface. If any regulus or speiss forms during the first fusion it must be preserved with special care and re-fused, charcoal being reduced or omitted. The button of lead from the second fusion is cupelled with the first button, or alone; the slag from the second fusion is almost always poor enough to be thrown away.

H. L. Sulman considers* that it is not necessary to pour and regrind the slag before "washing" it. Instead of this he uses a method similar to that adopted in "cleaning" scorification slag (see below, p. 463). When the crucible contents have fused and become quite tranquil, he stirs in first a little more oxide of lead and then a pinch of carbon, and thus makes a quick "washing" form part of the ordinary operation.

The Treatment of Base Ores.—The difficulty of roasting *arsenical and antimonial* ores may be avoided † by taking ore, 1 A.T.; red lead, 1,000 grains; sodium carbonate, 500 grains; potassium ferrocyanide, 550 grains, with a cover of salt or borax. The button is scorified, together with the matte formed, before cupellation. E. A. Smith ‡ found this method unsatisfactory and recommends the oxidation of sulphide of antimony by nitre, the charge being made up as follows in the case of an ore containing 75 per cent. of stibnite :—

Ore, ...	50 grms.
Red lead, ...	100 ,,
Sodium carbonate, ...	40 ,,
Borax, ...	10 ,,
Nitre, ...	30 to 40 ,,

With cover of powdered salt. The fusion is effected in large crucibles at a low temperature. Carbonate of soda forms a fluid slag with oxides of antimony. It is perhaps better to scorify these ores as well as those containing much copper if they are rich enough.

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* *Trans. Inst. of Mng. and Met.*, vol. ix. (1901), p. 355.
† Rickett's *Notes on Assaying*, New York, 1887, p. 77.
‡ *Trans. Inst. Mng. and Met.*, vol. ix. (1901), p. 334.

James recommends[*] that ores consisting chiefly of sulphide of antimony should be mixed with twice their weight of silica, and roasted at a very low temperature. The sand prevents the charge from "balling." Sulman prefers[†] to mix the charge with carbon instead of sand. In roasting this mixture in a reducing atmosphere, some 96 or 97 per cent. of the antimony is volatilised as sulphide at a fair red heat without loss of gold.

If they are poor, ores containing *copper* may be treated in three ways,[‡] so that each method may serve as a check on the others, viz. :—(*a*) Fusion with much PbO : the lead button becomes cupriferous, and should be scorified together with the matte; (*b*) roasting, followed by fusion and scorification; (*c*) treatment with nitric acid, by which all the sulphur and copper are removed. The silver dissolved in the liquid is then precipitated by a solution of common salt, of which a large excess should be avoided. The insoluble residue is dried, and can now be readily fused and cupelled. By treatment (*c*) the lead button is kept free from copper, the presence of which in the lead obtained by methods (*a*) and (*b*) renders cupellation difficult and unsatisfactory.

Zinc ores may be roasted, and fused with a somewhat larger quantity of borax and sodium carbonate than usual with pyritic ores. Instead of roasting, it is easier to desulphurise blende with metallic iron in the fusion, the zinc being volatilised.[§] *Lead* ores should not be roasted, but fused with plenty of iron. E. A. Smith[‖] recommends the direct fusion of *bismuth* ores with comparatively large quantities of sodium carbonate and borax, using a low temperature.

Telluride ores are not roasted but fused with excess of litharge in a moderately hot fire. Charges for two siliceous American ores containing tellurides are given as follows:[¶]—

	Poor Ore with little Pyrites.	Rich Ore with much Pyrites.
Ore, ...	1 A.T.	0·2 A.T.
Sodium carbonate, ...	1 ,,	1 ,,
Sand, ...	½ ,,	½ ,,
Litharge, ...	6 ,,	4 ,,
Argol, ...	2 grms.	1 grm.
Borax glass, ...	10 ,,	...
Common salt, ...	Cover.	Cover.

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[*] *Trans. Inst. of Mng. and Met.*, vol. ix. (1901), p. 353.

[†] *Idem*, p. 355.

[‡] Percy's *Metallurgy of Silver and Gold*, p. 247.

[§] E. A. Smith, *op. cit.*, p. 342.

[‖] *Op. cit.*, p. 346.

[¶] C. H. Fulton, *Journ. Amer. Chem. Soc.*, 1898, vol. xx., pp. 586-597.

The slag is remelted with 1 A.T. of litharge and 2 grammes of argol. The lead button is cupelled direct.

Tindall points out * that a most important point in the assay of telluride ores is to crush the ore finely. Cripple Creek ores are crushed through 120-mesh sieves if they are poor. Richer ores are crushed through 150-mesh sieves, and very rich ores through 200-mesh sieves. Lodge states † that in Cripple Creek a flux is used made up as follows:—

K_2CO_3,	3·07 kilos.	Mix well.
Na_2CO_3,	2·7 ,,	
Borax glass,	2·55 ,,	
Flour,	0·45 ,,	
Litharge,	13·6 ,,	

Take about 65 or 70 grammes—that is, 2⅓ A.T.—of this mixture to ½ A.T. of ore, and fuse in a muffle furnace. The slag will be glassy and brittle.

Hillebrand and Allen ‡ found that the crucible assay is perfectly satisfactory for telluride ores from Cripple Creek. The charge giving the best results on ores containing from 15 to 20 ozs. of gold per ton was as follows:—

Ore,	1 A.T.
Sodium bicarbonate,	1 ,,
Litharge,	6 ,,
Fused borax,	10 grms.
Salt,	Cover.

The losses of gold in the slag were generally small. The losses by cupel absorption were much more serious. On the other hand, E. C. Woodward found § that the cupel losses were small, never rising above 1 per cent. of the gold. J. C. Bailar records ‖ cupel losses ranging up to 100 per cent. Further investigation is evidently required.

For assay of tellurides by scorification see p. 463.

(2) **Cupellation.**—This operation is conducted in a muffle furnace, the construction of which is shown in Figs. 95-97, p. 474. The fire is lighted, a little bone-ash is sprinkled on the floor of the muffle to prevent its corrosion by litharge in case of the upsetting of a cupel, and the cupels introduced as soon as a bright red heat is attained. The cupels are cleaned by gentle rubbing or blowing before being charged-in, and are again cleaned with bellows before the lead is charged-in. They are placed in the furnace one by one, or, better, charged-in together on a tray, see below, p. 476.

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* *Trans. Inst. of Mng. and Met.*, vol. ix. (1901), p. 354.

† *Notes on Assaying*, by R. W. Lodge, New York, 1905, p. 131.

‡ *U.S. Geol. Survey*, series E (1905). Bulletin No. 253.

§ *Western Chemist and Metallurgist*, May, 1905.

‖ *Idem.*

Cupels are little cups made of bone-ash, and are either round or square. In their manufacture the bone-ash is finely powdered so that it will pass a 40-mesh sieve, then slightly moistened with water (to which a little carbonate of potash is sometimes added), put into a mould (Fig. 99), and compressed by the blows of a mallet, or, better still, of a screw press, so as to cohere firmly. The mould is preferably made of steel, as gun-metal wears sooner, and an uneven surface is disadvantageous. It is kept clean, bright, and smooth, and the sides should be tapered upwards slightly to facilitate the removal of the cupel without injury. Cupels must be dried very carefully and slowly, but completely, as otherwise cracks appear when they are heated, and loss is thereby occasioned. The cupels at the Royal Mint are made two years before being used, and are dried slowly on shelves at some distance from the furnaces. More rapid drying has been tried on many occasions with less satisfactory results. Nevertheless fair results, but not the best, can be obtained when using cupels only a few days' old. If too much water is used in mixing the bone-ash, the cupels lose part of their porosity; if too little is used they are too soft and crumble readily. About 6 or 7 per cent. of water answers very well. Care must be taken to preserve cupels from the access of nitrous or other acid fumes. These are absorbed by the bone-ash and given out in the furnace so that they may cause spitting. Lengthened heating of an hour or more in the muffle before charging-in the lead buttons will prevent loss from this cause. It is always advisable to heat cupels for 15 minutes after they have attained the temperature of the muffle before charging-in the lead buttons.

Bone-ash of good quality is supplied by several makers. It may be prepared * by calcining carefully clean knuckle-bones. The crude bone-ash is crushed through a 30-mesh sieve, and re-calcined. Caustic lime and carbonate of lime are next removed by lixiviation with ammonium chloride, and the bone-ash, after thorough washing and drying, is graded by sieving, and is then ready for use. The old German assayers of the 16th century used wood ashes mixed with bone-ash for all but the surface of their cupels.

Magnesia was introduced a few years ago, as a substitute for bone-ash. It is strongly calcined, and does not absorb acid fumes. The cupels require to be made under high pressure, and should be baked at a high temperature before use. They are then as hard and strong as fireclay. Magnesia absorbs and holds less litharge than bone-ash does. For the absorption of gold by these cupels, see below (p. 455).

The cupels having been sufficiently heated in the muffle, the lead buttons obtained as described on p. 455 are charged-in by the tongs (A, Fig. 94). The buttons collapse and lose their shape in a few seconds if the temperature is sufficiently high, but the molten mass formed is covered by a black crust for some time later. The crust then breaks up, and the brilliant surface of a liquid bath is seen. The muffle door should be closed immediately after the charging-in is completed, and kept closed until all the assays have thus "uncovered." The door is then opened a little way to let in a current of air by which the lead is oxidised, and the litharge, floating to the edge of the bath, is absorbed by the cupel, together with the oxides of other base metals, which are not taken up by bone-ash if they are in a state of purity. Uncombined metals are not readily absorbed by the cupel, and only traces of gold and silver are carried into it by the litharge (see below, p. 453).

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* Bettel, *Chem and Met. Soc. of S.A.*, vol. ii., p. 599.

As the cupellation advances the lead bath is reduced in size by oxidation and absorption; reddish patches float slowly over its surface, appearing earlier in the richer assays; it becomes more convex and brighter; the red spots move more quickly and finally whirl round with great speed and then disappear; moving iridescent bands take their place for a moment and then disappear likewise, and the bead becomes suddenly much duller in appearance, thus indicating that the cupellation is at an end. The cupels may then be removed from the muffle, provided the ore is poor or has little silver in it. It must, however, be remembered that silver absorbs oxygen when molten and gives it off suddenly when solidifying, so that if the bead weighs more than 0·01 gramme (Rivot) little fountains of metal are thrown up, and some part may be projected out of the cupel. This "sprouting," "spitting," or "vegetation" may take place in argentiferous-gold beads if the gold does not exceed one-third of the silver (Levol). At the Royal Mint it is found that a still larger percentage of gold does not prevent spitting unless a trace of copper is present. Where spitting is to be feared, therefore,

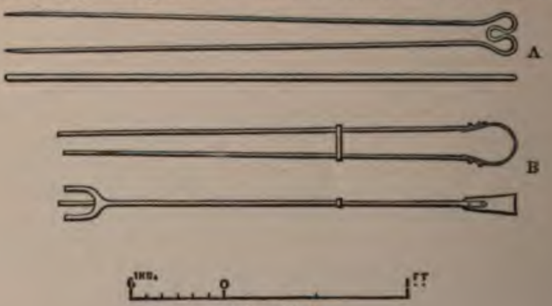

Fig. 94.*

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* Reproduced from Percy's *Metallurgy of Silver and Gold*, by kind permission of Mr. John Murray.

the cupel containing the silver bead is covered with a red-hot empty cupel, and is withdrawn gradually from the muffle so that the bead may cool slowly. Sprouting is avoided in this way. When the beads are very large (0·2 gramme or more) the muffle is closed and luted up and the fire allowed to go out, as in the assay of silver bullion.

When cooled the beads often "flash"—*i.e.*, brighten suddenly at the moment of solidification. This is due to the fact that the latent heat of fusion being released raises the temperature of the bead enormously, the metal having been in a state of surfusion at a temperature many degrees below its melting point. The flashing of small beads can rarely be observed.

The proper *temperature* for cupellation of gold ores is the same as for silver ores. The lowest temperature at which good soft lead uncovers is about 675°—that is, a dull red heat—and a good temperature for the air in the muffle is 700°, which need not be reduced throughout the operation. Litharge freezes at about 640°. When antimony, iron, &c., are present, the temperature must be much higher, and time is saved in all cases by using a higher temperature at first, but the losses are then greater. The muffle should be at a full red heat, the cupel dull red, and the melted lead much more luminous than the cupel; the fumes should rise slowly to the crown of the muffle. Many assayers prefer to finish the cupellation at a temperature low enough to enable crystals of litharge to form in a ring round the cupel—that is, at about 625°—but it is doubtful whether there is any advantage in this. The temperature must be rapidly reduced at the finish by bringing the cupel towards the front of the muffle, and even then the feathers are not readily formed unless the cupel is almost saturated with litharge. It is probably better that the litharge should be completely absorbed. The formation of scales, due to low temperature, is accompanied by a sluggish heavy movement of the fumes, which fall in the muffle. On the other hand, "the heat is too great when the cupels are whitish," the fused metal is seen with difficulty, and the scarcely visible fumes rise rapidly in the muffle (Mitchell). This statement is correct as far as it goes, but the temperature may be too low for these characteristics to be observable, and yet too high for cupellation. If the assays are long in uncovering they may sometimes be started by dropping on them a little charcoal powder wrapped in tissue paper, or, still better, by placing a piece of charcoal near them. If one freezes before completion it is restarted in the same way, or fresh lead is added or the temperature is raised, but the results are not good. Spirting or "spitting" of the lead may occur, due to the use of raw cupels, or to the presence of sulphur, arsenic, &c., in the lead.

The bead thus obtained should be well rounded and bright, loosely adherent to the cupel and slightly crystalline although malleable. If it contains lead it is more globular and brittle and its surface is very brilliant, while it does not adhere at all to the bone-ash. If copper is present in large quantities, the bead adheres firmly to the cupel, and in extreme cases its surface is blackened. Rhodium and iridium occasion black patches at the bottom of the bead; platinum makes the surface of the bead crystalline and rugose. If the bead cracks on being flattened Van Riemsdijk recommends a second cupellation with some more lead and 10 per cent. of cupric chloride. In this way the metals with volatile chlorides are eliminated.

*Influence of Base Metals on Cupellation**—*Iron.*—Its oxide is not readily fusible with lead oxide: the button is long in melting, and a brown scoria is sometimes left on the cupel, which may entangle lead globules and so contain gold. The lead may contain about 4 per cent. of iron, without the formation of any scoria. The cupel is stained dark red, and is moderately corroded.

Zinc burns with a brilliant green flame at first and volatilises; it forms a voluminous white or pale yellow scoria. The cupel is deeply corroded.

Tin gives a brown scoria if it constitutes more than 3 per cent. of the lead button. In all cases it floats to the top, oxidises, and forms a floating scoria which delays cupellation, but is subsequently carried off by the litharge. The cupel is stained pale brown.

Copper carries gold into the cupel and is usually not wholly removed from the bead. The cupel is stained dark brown or black.

Nickel and Cobalt are not so easily carried into the cupel as copper: they form a black scoria.

Antimony does not interfere if less than 2 per cent. is present. If 4 per cent. is present, a slight yellow scoria is formed, and the cupel is stained dark brown.

Arsenic at first burns with a blue flame and causes spitting. After a few minutes the flame turns greenish-white, and becomes very brilliant. The cupel is stained pale brown, and if 4 per cent. of arsenic is present, there is much pale brown scoria.

Manganese causes black stains and deep corrosion of the cupel, and forms a black scoria. The lead boils and spits.

Chromium gives a brick-red stain and a black scoria on the cupel, and *aluminium* a grey scoria; both these metals delay the course of cupellation. Chromium causes deep corrosion.

Cadmium causes a black sooty ring to form inside the cupel near its margin. When cupellation begins, the action is very violent, a yellowish-red flame is seen, followed by much spitting and the appearance of large red-hot scales floating on the lead. Oxide of cadmium is not absorbed by the cupel.

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* See "Notes on Cupellation and Parting," by T. K. Rose, *Chem. Met. and Mng. Soc. of S.A.*, Jan. 1905.

Tellurium is absorbed quietly, but whitish fumes appear, which give the bath a greenish-white appearance. The cupelled bead is sometimes, but not often, subdivided. The cupel has a brown stain in a ring round the litharge stain, but there is no scoria.

Selenium gives brown fumes, which change to blue fumes at a little distance. The effect on the cupel is similar to that of tellurium, but the brown stain is less pronounced.

Molybdenum causes slight boiling of the lead. The cupel is stained pale brown, and very deeply corroded, but there is no scoria.

Vanadium gives a dark red stain, and causes remarkably deep corrosion of the cupel, but leaves no scoria.

Thallium gives a pale brown, and *bismuth* an orange stain. Neither of these metals leaves a scoria.

Silicon gives a voluminous yellow or white scoria, which delays cupellation, and is left on the cupel.

Losses in Cupellation.—The loss during cupellation by volatilisation was proved to occur by Makins, but is insignificant. The absorption by the cupel is more serious. Rivot states that gold is oxidised to some extent at a red heat in the presence of antimonic oxide, litharge, or cupric oxide, and that it is the oxidised part which is absorbed by the cupel.

Still heavier losses may be caused by large amounts of scoria, in which the whole of the gold and silver may be retained in the form of minute beads, impossible to collect except by scorification with fresh lead. Small losses are sometimes caused by spitting, but these are usually trifling, as the spitting ceases before the lead bath becomes rich in gold and silver.

In some experiments made by the author,* the exact losses due to the presence of base metals were determined. In each case 25 grammes of lead were cupelled with 1 milligramme of gold, 4 milligrammes of silver, and 1 gramme of the impurity. Bone-ash cupels made at the Royal Mint were used. The results were as follows:—

Nickel and chromium caused total loss in each case, due to scoria.

Silicon occasioned losses amounting to between 40 and 50 per cent., due to scoria.

		Per cent.		Per cent.	
Tin, . .	Loss, gold,	2·0;	silver,	13·9.	Slight ring of scoria.
Zinc, . .	,,	9·3	,,	17·6.	Much scoria.
Cadmium, .	,,	3·5	,,	13·1.	Ring of black feathery scoria.
Arsenic, .	,,	3·9	,,	16·3.	Much scoria.
Antimony, .	,,	5·3	,,	13·2.	No scoria.
Manganese,	,,	13·6	,,	24·3.	Scoria.

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* *Loc. cit.*

		Per cent.		Per cent.	
Iron, . .	Loss, gold,	4·0;	silver,	16·6.	No scoria [illegible]
Copper, .	”	10·0	”	32·6.	” [illegible]
Bismuth, .	”	21·8	”	27·9.	” [illegible]
Thallium, .	”	23·1	”	34·4.	” [illegible]
Molybdenum,	”	11·0	”	26·2.	” [illegible]
Vanadium,	”	7·7	”	21·7.	[illegible]
Tellurium,	”	55·8	”	67·9.	[illegible]
Selenium, .	”	54·1	”	64·5.	[illegible]

The losses in cupellation under the same [illegible] temperature, air supply, &c., and using similar [illegible] pure instead of impure lead, were 1·2 per [illegible] per cent. of silver. In no case was [illegible] by volatilisation. Practically the whole [illegible] and silver was recovered by fusing [illegible] mg. and selenium were present. The [illegible] smaller these experiments appear very large [illegible] that the absolute losses were very [illegible] mg. silver that the formation of a scoria [illegible] acid by a heavy loss of gold.

The percentage losses are [illegible] larger. Thus E. A. Smith* found [illegible] mgs. of silver (= 324 mgs.) when [illegible] mg. of was 4·6 per cent., against [illegible] or partially substituted for bismuth, [illegible] probable, however, cent. on the crucible assay [illegible] by dropping silver per ton.

Lodge records‡ that [illegible] porcelain crucible if gold with 10 gramme [illegible] Nitric acid of to 2·99 per cent. at [illegible] parts water to four copper increased the [illegible] for use with all cent. In these [illegible] more dilute acids without being part[illegible] acid from chlorine

The absorption [illegible] acid containing some with the quality [illegible] must be previously the impurities [illegible] that case break up in cupelling small [illegible] is used. The acid Carter|| found [illegible] turning it black and set of cupels. [illegible] containing only a silver at about [illegible] a few seconds, and samples of [illegible] If the amount [illegible] for some minutes. [illegible] gold and 40 mgs. [illegible] residue washed twice

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[illegible] previous work on the [illegible] the crucible or to the

[illegible], 1905, p. 256.

Charges.	Losses per Cent. in Cupellation.		
	Magnesia Cupels.	Bone-ash Cupels, "A."	Bone-ash Cupels, "B."
Gold, 0·001 gramme, . Silver, 0·004 ,, . Lead, 25 grammes, .	Gold, 1·9 Silver, 8·3	1·2 12·2	7·8 27·1
Gold, 0·001 gramme, . Silver, 0·008 ,, . Lead, 10 grammes, .	Gold, 1·4	0·9	5·2
Gold, 0·5 gramme . Silver, 1·25 grammes, . Lead, 4 ,, .	Gold, 0·060	0·055	...
Silver, 0·5 gramme, . Lead, 4 grammes, .	Silver, 1·21	0·99	...

The absorption by magnesia cupels, according to these tests, is not much greater than that by bone-ash. Whatever cupels are used, the absorption should be tested frequently.

Losses of gold, and especially of silver, are greater at a high temperature than at a low one. This is illustrated by comparing the following mean results of experiments, in which the bone-ash cupels "A" were used, with the results in the table given above :—

Charges.	Losses in Cupellation at 700°.
Gold, 0·001 gramme, Silver, 0·006 ,, Lead, 25 grammes,	Gold, 0·45 per cent. Silver, 8·6 ,,
Gold, 0·001 gramme, Silver, 0·010 ,, Lead, 25 grammes,	Gold, 0·39 ,, Silver, 4·5 ,,
Gold, 0·010 gramme, Silver, 0·080 ,, Lead, 25 grammes,	Gold, 0·074 ,, Silver, 2·5 ,,

(3) **Inquartation and Parting.**—The bead of silver and gold obtained by cupellation is squeezed between pliers, or flattened by a hammer on a clean anvil, to loosen the bone ash adhering to its lower surface, and is then cleaned by a brush of wires or stiff bristles. It is then weighed, the silver removed by solution in nitric acid, and the weight of the residual gold taken, when the difference between the two weighings represents the silver. If the bead contains more than one-fourth its weight of gold, more silver is added to it as otherwise some of the silver will remain undissolved, being protected from the action of the acid

by the outer layers of gold. The amount of silver to be added is calculated from the (approximately) known composition of the bead, or guessed from its colour. A pale yellow bead always contains more than 60 per cent. of gold, but a perfectly white bead may not "part" completely. The addition of the silver is sometimes effected in the case of small beads by fusion on charcoal by the blowpipe, but it is better to cupel the bead with the additional silver, wrapped in as small a piece of lead foil as possible. This is called "*inquartation.*" The resulting bead is cleaned, flattened by a hammer, and dropped into nitric acid.

Even if the gold is less than one-fourth the weight of the bead, additional silver is sometimes required. When the amount of gold is small, it is convenient to use a large proportion of silver in parting. For example, if the gold weighs 0·2 mg., ten times its weight of silver answers very well. With smaller beads a greater proportion of silver is desirable. For 1 mg. gold, use 6 mg. silver. For 10 mg. gold, use 40 mg. silver. With these proportions, the gold does not break up if hot acid is used. If less silver is added, parting is not complete except after prolonged (20 to 30 minutes) boiling in acid.*

A. Whitby considers † that for 1 mg. of gold about 4 mgs. of silver should be used, and for 0·1 mg. of gold about 1·2 mg. of silver. He states that the beads break up entirely or partially if larger amounts of silver are used. It is probable, however, that he is referring to parting which has been begun by dropping the beads into cold acid.

The *parting* is effected in a test tube or a porcelain crucible if the bead is small, in a "parting flask" if large. Nitric acid of specific gravity 1·25 (made by adding three parts water to four parts of ordinary strong acid) is suitable for use with all beads. There is no advantage in using more dilute acids for poorer alloys. The freedom of the nitric acid from chlorine is ensured by adding to it a little spent acid containing some nitrate of silver. In all cases the acids must be previously heated to boiling, as the gold does not in that case break up into such fine particles as if cold acid is used. The acid attacks the bead instantly and violently, turning it black and giving off nitrous fumes. Little beads containing only a small proportion of gold are dissolved in a few seconds, and decanting may be at once proceeded with. If the amount of gold is large, the boiling is continued for some minutes. Ten minutes boiling is enough for 10 mgs. gold and 40 mgs. silver. The acid is then poured off, the residue washed twice with boiling water by decantation, and if the bead is very large fresh acid is added of specific gravity 1·30 (two parts of strong nitric acid to one part of water). The boiling is then continued for five to ten minutes longer, when almost all the silver will have been dissolved. A very small amount of silver, weighing from 0·05 to 0·1 per cent. of the gold, obstinately resists the action of the acid, and remains as a surcharge which may be neglected in almost all ores. The gold usually remains as a single piece if it weighs less than 0·1 mg., even if the proportion is only one part of gold to 40 or 50 parts of silver. If the ore is rich, the gold sometimes, though rarely, breaks up if hot acid is used, and invariably breaks up if the parting is begun with cold acid. The finer particles may float and be lost in decantation. Particles floating on the surface may be sunk by touching with a glass rod or by a drop of water let fall on them. Continued heating of the gold in acid makes it agglomerate to some extent, so that it is easier to wash.

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* *Loc. cit.* If the addition of silver is known by previous work on the ore to be necessary, it may be added to the charge in the crucible or to the lead during the original cupellation.

† *Journ. Chem. Met. and Mng. Soc. of S.A.*, April, 1905, p. 256.

After the final decantation of the acid and washing twice with hot water, the water is drained off and the porcelain crucibles dried at a gentle heat, and then gradually raised to dull redness. The gold which was previously black and soft, being in a fine state of division, now resumes its usual yellow colour, and hardens so that it can be removed to the pan of a balance and weighed. At a bright red heat the glaze on the porcelain crucible softens and the gold sticks to it. Excessive heating must therefore be avoided. An ordinary Bunsen flame answers very well.

If a parting flask or test tube is used for the boiling, the parted gold is transferred to an unglazed Wedgwood crucible. To effect this the flask is filled with water, and the crucible placed over its mouth. On inverting both together the gold falls into the crucible, and the flask is removed in such a way as not to disturb the precious metal. The water which has filled the crucible is then poured off, and the crucible heated as before.

The chief difficulty in parting by the last-named method is encountered in transferring the gold from the glass vessel to the Wedgwood crucible; minute particles of gold may adhere to the glass and are then left behind. The loss of these is the cause of the low results occasionally observed when this method is used. If the glazed porcelain crucibles are used for both boiling and annealing, no transference from one vessel to another of the gold in its soft state is necessary, so that the source of error mentioned above is avoided. On the other hand, the difficulty of boiling the acid in a small porcelain crucible without sustaining loss by projection may prevent the assayer from continuing the boiling for a sufficient length of time, and some silver may thus be left undissolved; in consequence of this the results obtained are sometimes too high by as much as 2 or 3 per cent. of the weight of the gold. Such errors would, however, be inappreciable in the assay of poor ores, and are the less serious for the reason that all the other sources of error (unclean slags, absorption by cupel, &c.) tend to make the result too low. The danger of loss by projection is avoided by using watch-glass covers for the crucibles. The latter are heated on clean asbestos sheets, not on sand baths. By assaying an accumulated stock of parted gold derived from ores, it was found at the Royal Mint by J. Phelps that the amount of silver retained by the gold when parted in crucibles as described above (p. 456), is about 0·5 per cent.

To prevent "bumping," one or two small pieces of capillary glass tube or of clay pipe or other porous body are sometimes put into the acid together with the alloy to be parted, but, as a general rule, it may be assumed that parting is finished when bumping begins.

Whatever method of parting is used, very finely-divided gold may remain suspended in the liquids, and may thus be lost in the course of decantation. Loss by decantation after parting may be avoided by adding a globule of mercury free from gold, and stirring with a glass rod until all the black particles of gold have been absorbed. The spent acid is then poured off and fresh nitric acid of density 1·2 added. On gently warming, the mercury is slowly dissolved and the gold remains as a coherent spongy mass. It is washed with nitric acid and then with water, glowed, re-heated with concentrated nitric acid, washed, re-heated, and weighed.

By the operation of parting, silver, palladium, and some platinum are removed in solution, but part of the platinum, and all the rhodium, iridium, &c., remain with the gold. If the presence of these metals is suspected they must be looked for and removed by special methods (*vide infra*, p. 493).

(4) **Weighing the Gold.**—The balance should turn with $\frac{1}{200}$ milligramme at most, and there is an advantage in using still more sensitive balances. A unit of 0·005 mg. corresponds to 2·4 grains per ton if 1 A.T. of ore has been taken, and 0·2 grain per ton if 12 A.T. have been taken. Great care must be exercised in placing the balance, to avoid vibration from machinery, traffic, &c. In certain cases the foundation must be laid at a depth of many feet and a column built for the support of the balance. The column should then not be allowed to touch any part of the superstructure. Balances are made with more care than weights, which are almost always inaccurate when received from the manufacturer. Small weights often differ from their stated value by 5 per cent., or even more. For this reason riders of 5 mgs. and 0·5 mg. should be used, the smallest weight used in the pan being 5 mgs. The errors on riders are

reduced according to their position on the beam. At a distance of one-tenth of the half-beam length from the centre, the error of the rider is reduced to one-tenth of its full amount. The rider must fit the beam so that it always lies in a plane perpendicular to it. The value of each division on the ivory scale is made, as nearly as possible, equal to 0·1 mg., 0·05 mg., 0·01 mg., or 0·005 mg., according to the sensitiveness of the balance, and the divisions are divided into 10 parts by the eye with the aid of a lens. The rider is always placed exactly on a subdividing mark on the beam, which should be nicked.

If riders are used, as suggested above, the difficulties due to errors in weights are reduced; but, if great accuracy is required, the riders must be examined, and, if necessary, adjusted by the assayer. It is necessary that the riders should be in concordance with the assay-ton weights. It may generally be assumed that the assay-ton weights and the 1 grm. or 0·5 grm. bullion assay weight are in agreement when received from the manufacturer. The errors on weights of 1 A.T. (29·166 grms.) or of 1 grm. rarely exceed 0·01 per cent. The assumption that these weights are correct involves a possible error of only 0·02 per cent., and this corresponds to only 0·1 grain of gold per ton on a 20-dwt. ore, and 0·01 grain of gold per ton on a 2-dwt. ore, amounts which are inappreciable. It is enough, therefore, to see that the rider bears the correct ratio to the gramme weight. This must be ascertained by building up a set of weights from 1 mg. to 1,000 mgs.

The weighings will be sufficiently accurate if they are made on a balance in which the divisions on the ivory scale are each equal to 0·005 mg. These divisions are divided into ten parts by the eye. Care must be taken to avoid unequal heating of the balance case. This can be watched if two thermometers reading to 0·01° C. are kept in the case—one near each end of the beam. In reading the indications of the balance, parallax is avoided by applying the eye to a pin-hole in a card fixed at about 12 inches from the ivory scale. A swinging ivory scale fixed to the beam and passing across a thread in a telescope gives still more exact readings. The position of rest of the balance is given by the formula, $\frac{l_1 + 2l_2 + l_3}{4}$, or, with greater exactness, by $\frac{l_1 + 3l_2 + 3l_3 + l_4}{8}$, where l_1, l_2, l_3, l_4 are successive positions of arrest (end of swing) of the balance, l_1 and l_3, on one side of the middle of the scale, and l_2 and l_4 on the other. If the middle of the scale is taken as zero, l_2 and l_4 will be negative.

The riders must be of platinum or aluminium. Brass riders increase in weight, and gilded brass riders oxidise and increase in weight still more rapidly. Some gilded brass riders at the Royal Mint increased in weight from 5·0 mgs. to 5·025 mgs. in six months. Platinum riders, if too light, may be gilded in a cyanide bath, but light aluminium riders must be rejected. Any rider may be reduced in weight by light rubbing on a sheet of ground glass, or by other mechanical means. The error on a rider can in this way be made less than 0·005 mg. without difficulty, but, if greater accuracy is desired, it saves time to determine the error of the weight and to apply it as a correction to the result of each assay.

The gold can be weighed by the method of substitution by which the bias of the balance is eliminated. The principle of this method is to counterpoise the gold, then to remove it from the pan, and compensate for the loss of weight by a rider. If direct weighing is preferred, the position of rest of the balance when the pans are empty is determined, the gold is put in the pan, a rider put on the beam in such a position as to approximately equalise the effect of the weight of the gold, and then the position of rest again determined. The method of swings is always used to determine the position of rest, disregarding the first swings. An example is appended.

A. *Position of Rest of Empty Balance.*—The swings, after the first two, are in succession + 3·4 divisions of the index, − 2·7 divisions, + 3·2 divs. (Plus divisions are always reckoned on the side where the weights are placed, minus divisions on the side where the gold is added.) The position of rest or true zero is $\frac{1}{2}\left(\frac{3 \cdot 4 + 3 \cdot 2}{2} - 2 \cdot 7\right) = +0 \cdot 3$ div. This determination is repeated after the parted gold assay pieces have all been weighed.

B. *Value of the Division.*—A weight of, say, 0·1 mg. (by means of a rider) is placed on the weight side, and the previous work repeated. The swings are − 2·1, + 0·4, − 2·0, and the new position of rest is $\frac{1}{2}\left(+0 \cdot 4 - \frac{2 \cdot 1 + 2 \cdot 0}{2}\right) = -0 \cdot 8$, and 1·1 division = 0·1 milligramme. This value does not alter much from day to day.

C. *Weight of Parted Gold.*—The gold is found to require about 1 milligramme to counterbalance it, and the swings are now − 4·6 + 0·2 − 4·4. The new position of rest is $\frac{1}{2}\left(0 \cdot 2 - \frac{4 \cdot 6 + 4 \cdot 4}{2}\right) = -2 \cdot 15$. The gold therefore weighs $1 - \left(2 \cdot 45 \times \frac{0 \cdot 1}{1 \cdot 1}\right)$ milligramme—that is, 0·78 milligramme.

By weighing in this way results correct to 0·002 mg. are easily obtained by using the best assay balances in which the value of the scale division is 0·005 mg., and still greater accuracy is attainable by weighing the gold in each pan in succession and by taking means of a number of observations.

Examination of Assay Materials.—The fluxes are usually free from the precious metals, but the litharge, red-lead, and granulated lead contain a small amount of silver and less gold (see p. 441). These are estimated by fusion with charcoal (or in the case of granulated lead by scorification) and cupellation. To prevent errors due to "salting," which may occur accidentally in an office where gold bullion or residues are handled, it is advisable to run a full blank charge at least once a week.

Examination of the Cupel.—When rich ores are assayed, appreciable quantities of gold are carried into the cupel, especially if certain base metals (see above, p. 452) are present in the lead button. To assay the cupel, all clean bone-ash is detached and thrown away, and the remainder crushed so as to pass through an 80-mesh sieve, and the charge made up as follows :—

Cupel, ...	100 parts.
Fluorspar, ...	75 ,,
Sand, ...	75 ,,
Sodium carbonate, . . .	100 ,,
Borax, ...	50 ,,
Litharge, ...	50 ,,
Charcoal, ...	4 ,,

Salt cover and hoop iron.

Or, according to Lodge,*

Cupel, ...	30 parts.
Litharge, ...	50 ,,
Borax glass, ...	20 ,,
Sand, ...	5 ,,
Argol, ...	2½ ,,

The fusion is made in an earthen crucible, and the resulting button of lead cupelled. The slag of the original fusion may be cleaned by adding it to the charge, when less fluxes will be required.

ASSAY BY SCORIFICATION.

As stated above, this process is especially applicable to (*a*) complex ores, (*b*) very rich ores, (*c*) ores which are mainly valuable for their silver contents. The losses are small and the operations are easy to conduct, and need not be varied much for different classes of ore. For these reasons the process is preferred to the crucible process by some assayers; if the ore is poor several assays are made and the lead buttons scorified together. The chief disadvantage of the process lies in the small quantity of ore that can be treated, so that the presence of one or two metallic particles of gold may cause the result to be erroneous.

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* *Notes on Assaying.*

Scorification is conducted in a muffle at a much higher temperature than that required for cupellation. It must be high enough to melt litharge when contaminated by silica and oxides of copper, iron, manganese, &c. A temperature of 1,050° to 1,100° C. is usually enough. The charge is placed in a *scorifier*, a shallow circular fireclay dish 2 to 3 inches in diameter, which is charged in by the tongs shown at B, Fig. 94. The charge consists of about 50 grains of ore (or $\frac{1}{10}$ A.T.), 500 to 1,000 grains of granulated lead, and a few grains of borax glass.[*] The ore is mixed with half the granulated lead, the mixture put in the scorifier and smoothed down, the rest of the lead spread over evenly and the borax put on the top. An addition of some litharge to the cover is often made. The amount of lead to be used varies with the nature of the ore. Ricketts and Miller give the following table [†] as a guide :—

Character of Gangue.	For one part Ore take	
	Parts of Granulated Lead.	Parts of Borax Glass.
Quartzose, . .	8 to 10	0
Basic, . . .	8 to 10	0·25 to 1·00
Galena, . .	6	0·15
Arsenical, . .	16	0·10 to 0·50
Antimonial, .	16	0·10 to 1·00
Fahlerz, . .	12 to 16	0·10 to 0·15
Iron pyrites, .	10 to 15	0·10 to 0·20
Blende, . .	10 to 15	0·10 to 0·20
Telluride, . .	16 to 18 and a cover of litharge.	0·10

The borax lessens the corrosion of the scorifier and renders the slag more liquid, but its quantity is kept as low as possible to prevent the slag from completely covering the bath of metal too soon. After charging-in, the door of the muffle is closed until fusion takes place. As soon as the lead is melted the door is opened, and a current of air allowed to pass over the bath of metal. Some of the ore is now seen to be floating on the surface of the lead, and is rapidly oxidised, partly by the air and partly by the litharge which immediately begins to form. The sulphur, arsenic, antimony, &c., are thus soon eliminated, while copper, iron, and other bases oxidise and slag off with the borax, and the silica and other acids form fusible compounds with the litharge. Effervescence and spirting may occur, especially if the scorifier has not been well dried by warming before it is used. The slag soon forms a ring completely encircling the bath of metal. As oxidation of lead proceeds, the litharge flows to the sides and increases the quantity of slag until at length the ring closes completely over the metal, leaving a flat uniform surface. This usually happens after from thirty to forty minutes. The slag should be "cleaned" before withdrawal. This is done by placing 3 grains of charcoal powder wrapped in tissue paper on the surface of the slag, with a pair of cupel tongs, and closing the muffle door. A number of globules of lead are formed by the reduction of the litharge, and these, falling through the slag, extract and carry down with them any gold and silver which it may still contain, and concentrate them in the molten lead below. The fusion being quiet again, the charge is poured into an iron mould by the scorifying tongs, and the lead button cleaned from slag by a hammer. If the button is too large to cupel at once it is re-scorified in the same dish, fresh lead being added if it is not soft and malleable.

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[*] By the use of a flatter scorifier than is usually made, or even by using a roasting dish, ½ A.T. of ore may be taken with a charge of 75 grammes of lead. E. H. Simonds, *Cal. Mines and Minerals*, 1899, p. 226.

[†] *Notes on Assaying*, 1897, p. 96.

The slag from the scorifier should always be separately retreated in the case of very rich ores, but seldom contains much gold.

The losses incurred in this process are chiefly due to improper temperatures. If the muffle is too cold at first, gold is retained by the slag. This initial low temperature is often indicated by the occurrence of white patches of sulphate of lead on the surface of the slag after pouring.* If the slag is pasty, borax is added, but the slag is then rich. It is better to begin again with less ore or more lead. When extremely rich sulphides are assayed, results are often obtained which do not agree well. Stetefeldt † recommends in all such cases that the sulphides be attacked by nitric acid and the residues dried and scorified.

Lodge states ‡ that the losses of gold in scorifying are greater than in cupelling. He accordingly recommends the use of the former method only when absolutely necessary, as in the assay of zinc residues from the cyanide process, and of copper ingots or bars and material rich in copper. No experiments seem to be on record showing the relative losses of gold when the crucible and scorification methods are used on ordinary quartzose or pyritic ores.

Tellurium ores often yield erroneous results, usually ascribed to volatilisation of the gold, but according to the author's experience, the vanished gold can be found in the slag or cupel, and has not been volatilised (see also p. 7). The tellurium must be driven off before cupellation, as otherwise the gold button may be subdivided into minute particles on the cupel, and in any case tellurium will carry much gold into the cupel.

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* Percy, *Metallurgy, Silver, and Gold*, p. 244.

† *Lixiviation of Silver Ores*, New York, p. 106.

‡ *Notes on Assaying*, p. 113.

Lodge* recommends the mixture in the scorifier to be covered with litharge when rich tellurides are being assayed. He considers the pot assay and scorification assay to be equally good for tellurides. Tindall† and Fulton are in favour of the crucible method.

Detection of Gold in Minerals.—The detection of gold in loose alluvial ground has been already described (p. 47). A similar method may be employed in the case of auriferous quartz after grinding it. If the concentrates obtained in either case contain sulphides, these are collected, roasted or treated by nitric acid and re-ground. The light particles of oxide of iron can now be separated from any gold that may be present by washing. "Colour" may often be obtained thus when none could be seen after the first concentration. The washing is made easier by removing from the concentrates the magnetic oxides and iron from the grinding tools by a magnet. All the finest particles of gold are lost in the process of washing, and consequently many auriferous ores cannot be made to "show colour."

The following method devised by Wm. Skey, analyst to the Geological Survey of New Zealand, is said by him to give good results. The sample of ore is carefully roasted, then digested with an equal volume of an alcoholic solution of iodine for a length of time varying from twenty minutes to twelve hours, the longer time being allowed if the ore is poor. A piece of Swedish filter paper is then saturated with the clear supernatant liquid and afterwards burnt to an ash; if gold is present in the ore the ash is coloured purple, and the colouring matter can be quickly removed by bromine. This method is said to show the presence of as little as 2 dwts. gold per ton in certain ores, but is not uniformly successful. Bromine or chlorine may be substituted for iodine. A mixture of 5 to 10 parts of bromine with 100 of water may be used to 100 parts of ore. The ore must be fine enough to pass an 80-mesh sieve and should be re-ground after roasting. After leaving the mixture to stand for some hours with occasional stirring, the liquid is filtered and the excess of bromine evaporated from the clear solution, which may then be tested by stannous chloride. Dr. Don‡ found these methods defective, and was forced to use the ordinary crucible assay when examining material containing small amounts of gold, taking samples of 4·48 lbs. of ore. L. Wagoner § determines the weight of small beads by fusing them and measuring their diameter under the microscope. The error in reading was about 0·001 mm. or 6·36 per cent. on a gold bead weighing 0·001 mg.

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* *Mineral Industry*, vol. viii. (1899), p. 397.

† *Trans. Inst. Mng. and Met.*, vol. ix. (1901), p. 354.

‡ *Trans. Am. Inst. Mng. Eng.*, vol. xxvii. (1897), p. 565.

§ *Trans. Am. Inst. of Mng. Eng.*, Nov. 1901.

SPECIAL METHODS OF ASSAY.

1. **Amalgamation Assay.**—This is useful in determining the amount of gold present in the ore capable of being extracted by mercury. The sample of ore is pulverised fine enough to pass through a 60- or 80-mesh sieve, made into a paste with a little water, an equal weight of mercury added, and the whole ground in an iron mortar with a pestle for from two to four hours, small additional amounts of water or mercury being added from time to time according to the appearance of the triturated mass. The consistency of the mass must be such that the globules of mercury do not sink in it but are broken up into very small particles. A little sodium amalgam dissolved in the mercury prevents it from flouring. The grinding is continued until the particles of ore are all impalpably fine. A machine for the purpose, called the "arrastra" mortar, has a large pear-shaped muller loosely fitting the inside of the mortar, and capable of being revolved in it by means of a handle. Complete amalgamation is performed in this machine much more rapidly than in an ordinary mortar. When the operator judges that amalgamation is complete, enough water is added to reduce the mass to a thin pulp and stirring is continued for a few minutes to collect the mercury at the bottom. The contents of the mortar are then "washed down" in a pan, the mercury collected and distilled, and the residue, consisting of gold, silver and base metals, scorified with test-lead, cupelled, and parted. If it is not necessary to estimate the silver extraction, the mercury may be dissolved by nitric acid, and the gold cupelled and weighed.

The sample may of course be panned before amalgamation. In that case Charleton recommends* treatment in a weighed golden dish, the surface of which has been amalgamated.

2. **Assay by Chlorination.**—Plattner's method for the assay of roasted pyrites consists in placing the mineral moistened with water in a glass cylinder, 200-250 millimetres deep and 20-30 millimetres in diameter, and introducing a current of chlorine gas at the bottom. When the odour of chlorine is noticed above the ore, a cover is put on, the stream of chlorine stopped and the whole left for twenty-four hours, after which the reaction is complete if chlorine is still in excess. Boiling water is now run through the ore until all soluble salts have been washed out, and the gold contained in the solution is precipitated by ferrous sulphate, collected, cupelled, and parted. This method fails if more than about 20 per cent. of silver is present, as the chloride of silver formed encrusts the gold and protects it from the action of the chlorine.† Balling recommends the addition of common salt to dissolve the chloride of silver, but found that the telluride

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* *Trans. Inst. Mng. and Met.*, vol. ix. (1900), p. 69.

† See results of Hofman and Magnuson on p. 261.

ores of Nagyag yielded only 85 per cent. of their silver and 92 per cent. of their gold when successively treated with chlorine and sodium chloride.[*]

Another method is to place the completely roasted sample in a stoppered bottle with enough water to make the whole of the consistency of thin mud. The ore and water should together occupy about two-thirds of the bottle. Bleaching powder and a thin glass bulb filled with dilute sulphuric acid are then added and the bottle securely closed. As cork is attacked by chlorine, glass or vulcanite stoppers are better, and the screw-stoppered soda water bottles are most convenient; if corks are used they must be wired down. The bottle is then shaken so as to break the sulphuric acid bulb and mix its contents with the bleaching powder, when chlorine is evolved. The bottle is now left for several hours in a warm place, being shaken occasionally by hand to mix its contents. At the end of a period of eight to twelve hours the bottle is opened, and if excess of chlorine is still present the liquid is separated from the ore and the latter washed thoroughly by filtration or decantation. The liquid and washings, whether clear or muddy, are warmed to expel free chlorine, and an excess of ferrous sulphate is then added to them. The precipitate is collected, scorified with lead, and cupelled. In all cases it is better to keep the first liquid separate from the washings, which should be concentrated by evaporation, since, if this is not done, the precipitate of gold may be too fine to settle and will pass through filter-paper. Bromine may be used instead of the materials generating chlorine. The quantities of chemicals required will be such as are sufficient to generate a volume of chlorine equal to twice the capacity of the bottle used, or to make a solution of 2 per cent. of bromine in water. Only finely-divided gold comparatively free from silver is extracted by this method.

3. **Assay of Metallic Copper and of Copper Matte containing Gold and Silver.**—Two methods are in use—a furnace method and a mixed wet and dry method.[†]

(a) *Furnace Method.*—Ten scorifications are made each of 0·1 A.T. of the sample, 50 grammes test lead (using half as cover) and 1 gramme borax glass as cover. The lead buttons are cupelled separately and the beads weighed together, and parted. The cupels are re-fused in five lots of two each with 90 grammes of litharge, 50 grammes each of soda and borax, and 3 grammes argol. The results are higher than those obtained by the mixed method.

(b) *Mixed Wet and Dry Method.*—Weigh out 1 A.T. of the auriferous material, place it in a beaker of a litre capacity, and add

[*] Balling, *L'Art de l'Essayeur*, p. 433.

[†] A. H. Ledoux, *Am. Inst. Mng. Eng.*, Oct. 1894; and Peter's *Modern Copper Smelting*, 7th ed. (1895), p. 67.

gradually enough nitric acid of specific gravity 1·14 to dissolve it completely; heat until red fumes cease to come off, dilute to 600 c.c. with water, and add 10 c.c. of a concentrated solution of lead acetate and 5 c.c. of concentrated sulphuric acid, and allow the lead sulphate to settle. The precipitate is filtered off and washed. It contains the gold which has been collected and carried down by the sulphate of lead. The filter-paper and precipitate are dried, the paper burned, and the ash and lead sulphate scorified with test-lead. The button is cupelled and the gold with any trace of silver it may contain is weighed and then parted.

The silver is determined on a separate sample, which is dissolved and treated as before, except that a slight excess of sodium chloride, or, better still, of sodium bromide (Whitehead*) is added after the sulphuric acid and before the lead acetate. The solution is well stirred, and the precipitate allowed to settle and treated as before. The use of the lead acetate is to cause the precipitates of gold and silver to settle quickly, and to enable them to be filtered effectively. Sodium bromide is used instead of the chloride, on account of the greater insolubility of the silver salt.

4. **Assay of Purple of Cassius.**—One part of purple of Cassius is fused with three parts of carbonate of soda, cooled and dissolved in water. The gold remains undissolved, and is collected on a filter and cupelled after incineration.†

5. **Assay of Graphite Crucibles.**— Graphite crucibles, stirrers, &c., contain considerable quantities of gold and silver after use in melting bullion. They may be crushed with water by edge-runners in mills resembling mortar mills or Chilian mills (*q.v.*), and the products washed down to separate shots of metal. The residues, even after roasting, yield only a moderate proportion of their values when treated by amalgamation, and can still be profitably smelted. The sampling of the crushed material before amalgamation is difficult, and must be carried out with great care. The dried sample is roasted and scorified, or fused in pots at the Royal Mint, but Dr. Loewy obtains correct results by scorifying without previous roasting.‡ He takes 5 grammes of the material, 60 grammes of lead, and a little borax or glass powder. The charge at the Royal Mint is—

Roasted material, ...	100 grains.
Lead, ...	1,000 ,,
Borax, ...	10 ,,

Half the lead is used as a cover. The slag is cleaned in the usual way, and is subsequently remelted in a crucible and a mixture of litharge and charcoal thrown on to the surface of the fused material. In this way from 0·5 to 2·5 per cent. of the total values are recovered from the slag. The cupellation loss is usually from 0·5 to 1 per cent.

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* *Chem. News*, 1892, vol. lxvi., p. 19.

† *Ann. de Pharmacie*, vol. xxxix.

‡ *Proc. Chem. and Met. Soc. of S. A.*, vol. ii., p. 205 (Jan. 1898).

6. **Estimation of Gold in Dilute Solution.**—A. Carnot has shown * that the rose colour produced in presence of arseniate of iron is very sensitive, and can be used for colorimetric estimation of small quantities of gold. To attain this end a neutral solution of chloride of gold of known strength is prepared, and some drops of solution of arsenic acid added slowly to it. Then after a time two or three drops of dilute ferric chloride and some hydrochloric acid are added. If the liquid is not acidulated, a flocculent purple precipitate forms; if it is too acid the reaction fails, and only a faint blue colour is seen. The liquid is made up to 100 c.c. with distilled water, a pinch of zinc dust added, and the mixture shaken in a flask. A colour is produced, varying from rose to purple, according to the amount of gold present. The solution is clear, can be filtered unchanged, and kept without alteration for some time. If more than one milligramme of gold is present (1 in 100,000), the colour becomes too intense for small differences to be noticeable; if less than one-tenth of this amount is present (1 in 1,000,000), the colour becomes too faint. Between these proportions the amount of gold present in a liquid can be determined by comparison with a series of prepared coloured solutions.

For more dilute solutions of chloride of gold, the test described on p. 27, depending on the use of stannous chloride, may be used. A number of precipitates are prepared from solutions of gold containing known amounts, and compared with that given by the solution to be estimated. Suitable volumes of liquid from which the test precipitates are obtained are as follows:—

Strength of Solution. Parts of Water present to each part of Gold.	Volume of Solution used to obtain a Precipitate.
1 to 5 millions,	100 c.c.
5 ,, 20 ,,	500 ,,
20 ,, 50 ,,	1,000 ,,
50 ,, 100 ,,	3,000 ,,

If more gold is present than one part per million, the colour of the precipitate is too intense for accurate measurement.

Vanino and Seeman † recommend the reduction of gold chloride by hydrogen peroxide in the presence of caustic soda or potash. The reaction is complete in a few minutes in the cold. In dilute solutions, the hydrogen peroxide is destroyed by heating the solution, and hydrochloric acid is added to agglomerate the precipitate. A solution containing 3 parts of gold in a million gives a slight red colour with a bluish shimmer.

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* *Bull. de la Soc. Chimie*, 1883.

† *Berichte*, vol. xxxii. (1899), pp. 1968-1972; and *Journ. of the Chem. Soc.*, vol. lxxvi. (1899), part ii., p. 578.

The methods of estimation of gold in dilute solutions of cyanide are given on pp. 346 and 370.

7. **Assay by Electrolysis.**—If the gold is in solution, potassium cyanide is added in the proportion of 2·5 grammes per 100 c.c. of solution, and a current is passed through the solution between platinum electrodes. The current density should be about 0·3 ampere per square decimetre, the fall of potential 2·7 to 4 volts, and the temperature 50° to 60° C. The gold is deposited in about 1½ hours,* and is weighed together with the cathode in the usual way.

According to F. M. Perkin and W. C. Prebble † it is better to use a solution of ammonium thiocyanate instead of potassium cyanide. With a current of 0·4 to 0·5 ampere per square decimetre, deposition is complete in 1½ to 2 hours at ordinary temperatures. The best way of removing the deposited gold from platinum electrodes is by means of a solution of potassium cyanide containing an oxidising agent such as hydrogen peroxide, sodium peroxide, or an alkali persulphate.

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* Classen, *Quantitative Analysis by Electrolysis.* Translated by Boltwood, 1903, p. 207.

† *Electro-chemist*, Feb. 1904, pp. 490-494.

CHAPTER XX.

THE ASSAY OF GOLD BULLION.

Introduction.—The assay of gold bullion, as described in this chapter, has for its sole object the estimation of the percentage of gold present in the alloy, all other constituents being disregarded. In the first instance, the simple case of the assay of gold alloys containing appreciable quantities of only copper and silver will be dealt with. Refined gold ingots and the alloys used for coinage, and for almost all jewellery, come under this head. The effect of large quantities of other impurities and the precautions thereby rendered necessary will be discussed later.

The method universally employed is that of cupellation and subsequent parting. The gold bullion is cupelled with silver and lead, by which the greater part of the base metals present is removed as oxides dissolved in litharge, and an alloy of gold and silver left on the cupel. This is "parted" by nitric acid, which dissolves the silver and leaves the gold unattacked.

In the following pages the practice at the Royal Mint is described, but the same description would apply, with very slight alterations, to the methods used at other mints and assay offices.

The "parting assay" was first mentioned in a decree of King Philippe of Valois, published in the year 1343.[*] The methods of procedure in the 17th century have been briefly described by Savot[†] and by J. Reynolds,[‡] and more fully in the *Compleat Chymist.* In 1666 Pepys saw the parting assay being practised at the Mint in the Tower of London, and from his description it is clear that the method then employed bears a surprisingly strong resemblance to that of the present day. In 1829 a Royal Commission was appointed in France to examine into all questions relating to the methods of assaying gold and silver. The results of the labours of that Commission are to be found in great part in the Appendix to the *Report of the Select Committee on the Royal Mint,* 1837. The Committee arrived at the conclusion that the method adopted for assaying gold often overstated the amount of precious metal by 1 part per 1,000.[§] The

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[*] *First Report of the Royal Mint*, 1870, p. 103.

[†] *Discours sur les Medalles Antiques*, Paris, 1627, p. 72.

[‡] *A New Touchstone for Gold and Silver Wares*, London, 1679, p. 362.

[§] *Report on the Royal Mint*, 1837, Appendix B, p. 123.

Mint Conference held in Vienna in 1857,* resulted in the almost universal adoption of a more uniform method of manipulation.

The degree of accuracy now attained in most assay offices reduces the probable error in the report of an assay to about 0·1 per 1,000, but, to prevent the error from rising above this amount, all weighings must be correct to 0·05 per 1,000, which is not always the case in ordinary bullion assays.

The system may be conveniently regarded as comprising six distinct operations, viz. :—

1. Selection of the sample.
2. Preparation of the assay piece for cupellation.
3. Cupellation.
4. Preparation of the assay piece for parting.
5. Parting and annealing the cornets.
6. The final weighing and reporting.

1. **Selection of the Sample.**—Alloys of gold with either silver or copper or with both are practically uniform in composition if they have been melted and well mixed, and the ingot has not been pickled after casting. In these cases a single outside cut is representative of the composition of the whole of the ingot. The cut must, of course, be clean.

When other metals are present, the solidified ingot is not uniform in composition, and a dip-sample is taken. The metal is melted in a plumbago crucible with a covering of borax, stirred vigorously, and one or more dip-samples taken by an iron ladle, or better, by a plumbago spoon. The sample may also be taken by a plumbago stirrer, the foot of which has a hollow cavity,† or by a little plumbago crucible fastened by iron wire to an iron rod. The sample is dipped out with a borax cover, and poured into an iron mould or allowed to cool in the dipper. Granulation by pouring into water may result in partial oxidation of some of the base metals.

Moderately pure tough ingots are sometimes sampled by means of outside cuts, even if there is no certainty that some base metal besides copper is not present. Cuts from a top corner and an opposite bottom corner are then taken, or drillings are taken from the top and bottom of the ingot. The results are not always trustworthy (see p. 20).

The sampling of gold wares is difficult, because the outside is usually finer than the interior, in consequence of the pickling of the wares after manufacture. The sampling of "base bullion" belongs to the metallurgy of lead.

2. **Preparation of the Assay Piece for Cupellation.**—If

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* *Kunst- und Gewerbeblatt Baiern*, 1857, p. 151.

† See *Proc. of Chem. and Met. Soc. of S. A.*, vol. ii., p. 794. The use of iron tools is apparently not approved by the Committee appointed by this Society to consider the matter. Iron would absorb sulphur, &c., fro the sample.

necessary, the assay piece is "flatted" on a clean anvil by a hammer with a rounded face, weighing about 11 lbs., and a portion, weighing about 0·5 gramme, is obtained by cutting with shears and filing. The metal to be cut is held firmly between the fore-finger and thumb of the left hand, and care is taken to keep the plane of the piece of gold perpendicular to the cutting faces of the shears, otherwise damage is done to the latter. Only clean portions of metal must be used.

When the assay is reported to $\frac{1}{10000}$ part, it is evident that the balance used must clearly indicate a difference in weight of 0·1 per 1,000 or 0·05 milligramme. It is convenient to have the balance adjusted so that one subdivision on the ivory scale traversed by the pointer corresponds to this quantity. The weight of alloy must not differ from the 1,000 by more than four subdivisions of the scale, and the difference is noted down as the "weighing-in correction." The weighed piece is wrapped in pure lead foil together with the silver necessary for parting, and some copper unless it is present in the assay piece. The lead packets are put in order in the numbered compartments of the wooden tray shown at *c*, Fig. 101, p. 478, their position being noted on the assay paper. The corners of the packets are squeezed down so as to fit the cupels by pliers specially designed for the purpose, and the assays are then ready to be charged into the furnace.

It was formerly considered necessary for the metals to be present in the proportion of 1 of gold to 3 of silver, but, as early as the year 1627, Savot relates that the proportion of 1 to 2 was used, and strong acid employed in the boiling, "quand on veut faire quelque essay curieux et exact."* Both Chaudet and Kandelhardt recommend the proportion of 1 to 2½, on the ground that less silver is then retained by the cornet than in any other case. If much more than three parts of silver are present the gold breaks up in the acid. Pettenkofer† found that the proportion of 1 to 1·75 could be employed if the assays were boiled in concentrated nitric acid for some time. At the Royal Mint the proportion used was formerly 1 to 2·75, but is now 1 to 2 (see also p. 486 in section on "Surcharge"). The test silver should be assayed for gold, but the presence of a small quantity, say one part or less in 100,000, does not matter if pieces of the same silver are used for "proof" and ordinary assays.

The amount of lead used varies with the proportion of base metals present. For gold 900 fine and upwards, eight times its weight of lead is used at the Mint and answers very well, but the copper is not completely removed in the course of cupellation. The following table shows the proportions recommended by D'Arcet,* Cumenge & Fuchs,† and Kandelhardt‡ respectively:—

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* *Discours sur les Medalles Antiques*, Paris, 1627, p. 73.

† *Bergwerksfreund*, 1849, vol. xii., p. 6.

Gold in 1,000 parts: the Alloying Metal being Copper.	Amount of Lead employed for one part of Alloy.		
	D'Arcet.	Cumenge & Fuchs.	Kandelhardt.
1,000	1	1	8
900	10	14	16
800	16	20	20
700	22	24	24
600	24	28	24
500	26	32-34	28
400	34	32-34	28
300	34	32-34	32
200	34	32-34	32
100	34	30	32
50	34	28	32
0	...	11	32

Kandelhardt's table is modified to make it uniform with the others. It must be remembered that, although the above table gives the quantities of lead which will remove the greater part of the copper present in the alloy during cupellation, the last 2 or 3 millièmes are obstinately retained by the gold, and cannot be entirely eliminated even by a second cupellation with fresh lead. Instead of attempting to remove all the copper present in an alloy of low standard by one operation, using large quantities of lead as above, it saves time and gives more uniform results if a smaller amount of lead is used in two successive cupellations. For one part of gold bullion 400 fine, 16 parts of lead are enough if added in this way. The object in view is not to remove all the copper by cupellation, but to obtain a well-formed, clean and bright button suitable for parting. Some copper must be retained by the assay piece in order to prevent "sprouting" at the moment of solidification of the cupelled button. For this reason, the amount of copper in the assay piece charged into the furnace should be not less than 50 parts per 1,000 of gold.

3. **Cupellation.**—A coke muffle furnace still in use at the Royal Mint is shown in front and side elevation in Figs. 95 and 96. It consists of an outer casing of wrought-iron plates about $\frac{1}{8}$ inch thick, united by angle iron. This casing is connected with a chimney 60 feet high by means of a wrought-iron hood, *a*, and flue which is provided with a damper, *b*. The lining consists of Stourbridge firebricks. Fuel is introduced through the opening *c*.

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* Pelouze et Frémy, *Traité de chimie générale*.

† *Encyclopædie Chimique*, vol. iii., L'or, p. 154.

‡ *Gold-Probirverfahren*, Berlin, p. 3.

The muffle, *d*, is made of a mixture of graphite and fireclay which lasts better than fireclay alone. Its mouth is closed by the graphite block, *e*, and the sliding iron plate, *f*, when cupellation or scorification is going on. For annealing and other purposes the sliding doors, *g*, *g*, are used. Air is admitted to the muffle above the block, *e*, only, and is withdrawn through the graphite tube, *h*, which connects with the iron tube, *k*. This is furnished with a damper, *l*, and is connected with the chimney. There is no other opening in the muffle, so that the draught through it is quite independent of the draught through the furnace. This is an important feature which should be adopted in all muffles. The muffle rests on a bed of fireclay and pieces of firebrick, which covers the cast-iron girder-plate, *m*. The top of the muffle is covered with a thick layer of fireclay and graphite to check radiation and to preserve the muffle. The clearance between the muffle and the sides of the furnace is 3¼ inches, but at the back it is only 1 inch, and this space is filled with fireclay to prevent overheating. In the gas furnaces the muffle now fits closely against the back wall of the furnace, so that the back is not hotter than the front.

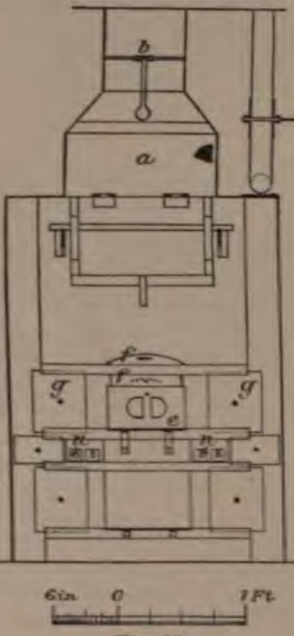


Fig. 95.

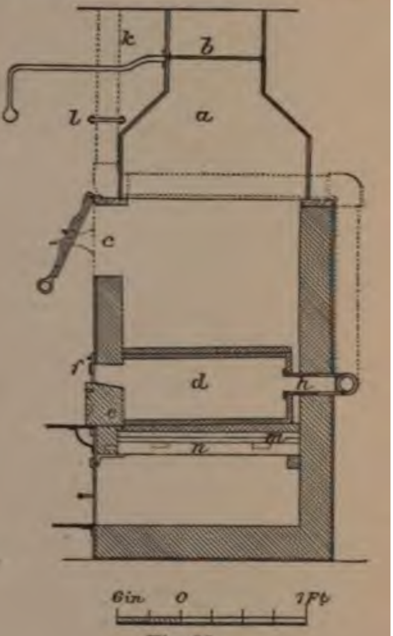


Fig. 96.

There are eleven fire-bars, *n*, *n*, but only the three outer ones on each side are covered with fuel. On the other five bars rests the cast-iron girder-plate (Fig. 97), which is flat on the upper surface, but is strengthened with ribs on the under surface in order to prevent buckling. Charcoal, anthracite, or coke can be used in this furnace.

The use of assay furnaces, heated by gas, is now general, wherever gas can be obtained. Gas furnaces, with separate draught to the muffle, as described above, are now preferred at the Royal Mint. Petroleum furnaces are used when coal, coke, and gas are alike difficult to obtain. Graphite muffles, though advantageous in coke muffles, do not seem to last so well as fireclay muffles in gas furnaces.

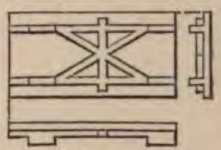
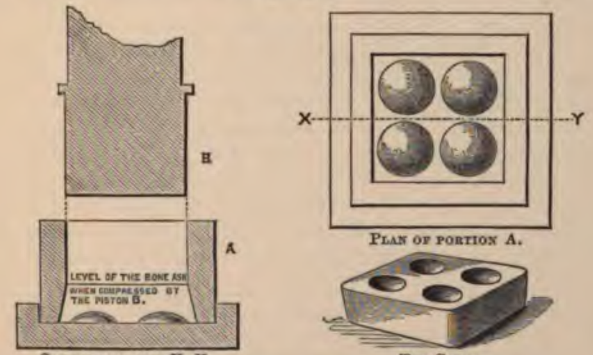


Fig. 97.*

Furnaces to burn soft coal are sometimes used in places where good coke is very expensive. They differ little from coke furnaces in construction, but have less space between the muffle and the side walls. The flame of the coal is chiefly instrumental in heating the muffle, a comparatively thin bed of fuel being employed.

The cupels in use at the Royal Mint are in sets of four, the outer margin being square, as shown in Fig. 98. At some assay offices, larger numbers of cupels are made in one block. The object is to facilitate charging-in and withdrawal, but the difficulty of maturing the cupels increases with the size of the block, and it is better to use cupel trays in the furnace. Single cupels can then be rapidly ranged on the tray by hand, before it is charged-in.



SECTION ON LINE X, Y.

PLAN OF PORTION A.

THE CUPEL.

Fig. 98.

The cupel tray used at the Royal Mint is shown in plan and section in Fig. 99. It is 11¾ inches long and 6 inches wide, and holds cupels for 72 assays, the cupels fitting tightly inside the rim. The tray is made of "Salamander" graphite, and lasts for some weeks. Iron trays were found to interfere with cupellation and to be rapidly destroyed. Fire-clay trays soon break. The tray is sprinkled with bone-ash, and the cupels placed on it before it is charged into the furnace. It is charged-in and withdrawn by an iron "peel" with two flat prongs, 1¼ inches wide, and of the same length as the tray, which slide into the grooves underneath the tray. The furnace tools formerly in use are shown in Fig. 100, where *a* represents the cupel tongs and *b* the tongs used for charging-in the lead packets. These tongs are now seldom used in bullion assaying.

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* Reproduced from Percy's *Metallurgy of Silver and Gold*, by kind permission of Mr. John Murray.

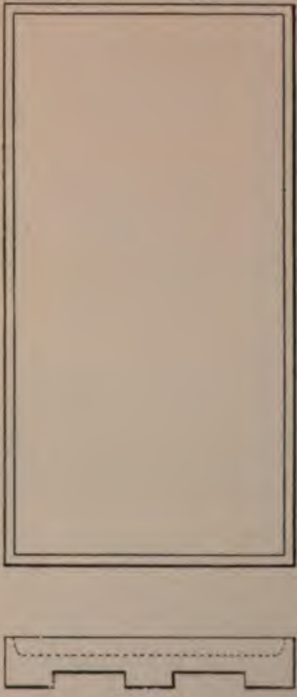
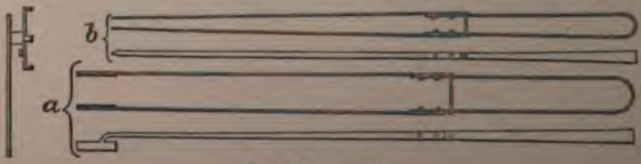


Fig. 99.

Method of Operation.—The muffle is brought to a uniform orange-red heat before the cupellation is begun and no fresh fuel is added during the operation. The cupels are cleaned by a pair of hand bellows just before the assay pieces are charged-in. This is now done by a charging tray which was devised by Mr. Henry Westwood, Assay Master of the Birmingham Assay Office, in 1893. It is made of nickel and consists of a plate perforated by 72 holes. Underneath is a non-perforated sliding plate. The assay pieces are put in order in the compartments, and the tray is placed in position, some help in guiding it being given by stops which touch the



Scale, 1 in. = 1 ft.
Fig. 100.*

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* Reproduced from Percy's *Metallurgy of Silver and Gold*, by kind permission of Mr. John Murray.

front cupels. The sliding plate is then withdrawn and the lead packets fall through the holes into the cupels. "Uncovering" is completed in about two minutes, and during this space of time the muffle is kept closed. The door is then opened, and the draught started by opening the damper (*l*, Fig. 96).

Distinct stages may be noted in the action which now takes place on the cupel. Almost immediately the surface of the molten metal becomes covered with greasy-looking drops of litharge, which are rapidly absorbed by the porous cupel and replaced by others. They pass over the surface at first slowly, but as the operation continues move with greater rapidity. In from twelve to fifteen minutes the metal suddenly becomes uniformly dull and glowing except for iridescent bands, produced by extremely thin films of fluid litharge, which are seen to pass over it. On the disappearance of these bands a bright liquid globule of a greenish tint is left, but the cupels are not withdrawn from the furnace until the expiration of another fifteen to twenty minutes so that the last traces of lead may be oxidised and absorbed. The completion of cupellation takes place first in the front rows and proceeds regularly backwards.

The cupels are withdrawn from the furnace while the assay pieces are still fluid, and "flashing" ensues in a few seconds. "Flashing" is most marked in the purer buttons, in which but little copper or lead remains. Slight effervescence may occur in these cases, but if not less than 50 parts of copper per 1,000 of gold were originally present the buttons are never sufficiently freed from base metals for "sprouting" to take place. If the cupels are withdrawn carelessly the fluid metal moves over the rough surface of the cupel, and minute beads may be separated from the main mass. To avoid this, and losses by sprouting, some assayers do not remove the assays from the furnace until the buttons have become solid.

The "flashing" of gold assays was shown by Van Riemsdijk * to be due to solidification after superfusion. The temperature of the fluid metal falls until a certain point is reached, when it solidifies, and the sudden disengagement of the latent heat of fusion reheats the cooling globule to its true melting point—viz., 950°—and a peculiarly intense light is emitted which rapidly fades as the temperature again falls. A sudden jar at any moment causes the flashing to occur instantly. If the alloy contains a minute quantity of iridium, rhodium, ruthenium, osmium or osmium-iridium (the metals of the platinum group which are non-malleable, refractory to heat, and resist the action of acids), the tendency of the cupelled metal to preserve its liquid state below the melting point, and therefore to flash during the final solidification is entirely prevented. The presence of metals of the platinum group (except platinum and palladium) in ingots of commercial gold can be detected by means of this characteristic.

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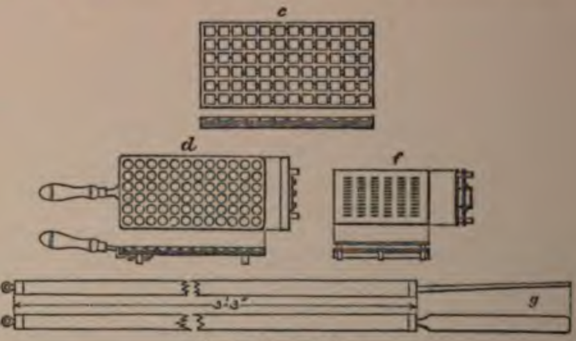
* *Chemical News*, vol. xli. (1879), p. 126.

The buttons (which are of the form represented at *a*, Fig. 102, p. 479) are removed from the cupels, after cooling, by a pair of sharp-nosed pliers, cleaned by means of a stiff brush or by immersion in warm dilute hydrochloric acid, and are placed in the compartments corresponding to their cupels in the tray (*d*, Fig. 101). If the bone-ash is not completely removed from their lower surface it is of little moment, since bone-ash is readily dissolved by nitric acid on parting. The surface of the cupels must be carefully examined for minute beads of metal due to spirting of the lead bath, which sometimes happens if there is too strong a draught. If any such beads are found in a cupel the fact is noted and the assay repeated.



Scale, 1 in. = 18 ins.
Fig. 101.*

If traces of lead remain in the button it is more globular, separates more easily from the bone-ash of the cupel, and has a brilliant steely surface. The effect of the presence of other metals is discussed on p. 491.

Temperature.—The exact temperature suitable for cupellation can only be ascertained by practice, and the varying light of the day may occasion error in judging the degree of heat. The remarks on temperature in the cupellation of buttons from ores (p. 451) apply here. Care should be taken to ensure that the heat is so high before "charging-in" that the chilling which necessarily takes place during this operation shall not cool the muffle below the requisite temperature. It is of more consequence that the muffle should be uniformly hot throughout than that any absolute degree should be attained, as the checks used eliminate uniform errors due to high temperature.

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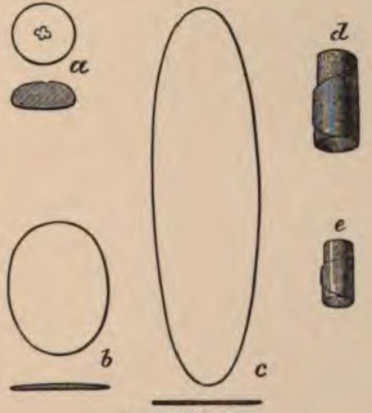
* Reproduced from Percy's *Metallurgy of Silver and Gold*, by kind permission of Mr. John Murray.

The temperature of an assay muffle was measured by Prinsep, Assay Master of the Mint at Benares, by observing differences in the behaviour of a number of silver-gold and gold-platinum alloys when heated.* He "made trials in different parts of the same (muffle) furnace. The disparity of heat," he remarks, "is greater than might be supposed." His results were as follows:—

	Maximum Alloy Melted.	Corresponding Temperature.†
Front of muffle, ...	pure silver.	960°.
Middle ,, (average), .	silver, 70. gold, 30.	1,025°.
Back ,, (,,), .	silver, 50. gold, 50.	1,045°.

In 1892, the temperature of a muffle at the Royal Mint was measured by the author by means of the Le Chatelier pyrometer.‡



Scale, full-size.
Fig. 102.

The muffle was 15 inches long and 6½ inches wide, and it was found that the temperature gradually rose from about 1,050° to 1,080° in passing from front to back, whilst along the sides the temperature was 1° or 2° higher than in the middle line. The mean temperature of the muffle was about 1,070°. It is believed that with gas furnaces and the present arrangements of the muffle the variation in temperature is much less.

4. **Preparation of the Assay Buttons for Parting.**—The buttons are flattened by a hammer weighing about 7 lbs., with a convex face, on an anvil kept quite bright and clean and used

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* *Phil. Trans. Roy. Soc.*, 1828, p. 79.

† Roberts-Austen and Rose, *Proc. Roy. Soc.*, vol. lxxi. (1902), p. 162.

‡ *Journ. Chem. Soc.*, vol. lxiii. (1893), p. 707.

for this purpose only. A heavy blow is first delivered on the centre of the button, the diameter of which is thereby increased to nearly that of a threepenny piece. Two lighter blows are then given on opposite sides of the disc so as to elongate it, giving it the form shown at *b*, Fig. 102.

After being annealed in the iron tray, *f*, Fig. 101, which is placed in the muffle by the tool, *g*, and left until it is red hot, the flattened buttons are passed in succession through a pair of jeweller's adjusting rolls which are used for this purpose only, and are kept clean and bright. The oil is removed as completely as possible from the rolls before they are used, as otherwise the first fillets come out thinner than the remainder. The rolls are adjusted so that one passage through them reduces the buttons to the required thickness, which is about 0·25 millimetre, or 0·01 inch, or about the thickness of an ordinary visiting card. The "fillets" (*c*, Fig. 102) thus obtained should all be of uniform size and thickness, with "wire edges," as ragged edges expose them to loss during the boiling. After being rolled they are replaced in the tray, *f*, and annealed at a dull red heat. The buttons and fillets are often annealed by a blowpipe on charcoal instead of in the furnace. They must not be made too hot, as that entails a loss of gold in the parting acid. It is best to raise them to a low red heat. The object of the first annealing is to soften the buttons and facilitate their passage through the rolls, while that of the second is to enable the fillets to be rolled into "cornets" or spirals, *d*, between the finger and thumb, or round a glass rod. Care is taken to leave that which was formerly the lower side of the button outside, for a reason given below (p. 495). This face is easily recognised, as it is less brilliant than the other.

5. **Parting.**—This was formerly effected by boiling with nitric acid in glass "parting flasks." Platinum boiling trays save time, and are now used whenever possible. The silver is dissolved by the acid, which must be free from chlorine in any form, sulphuric and sulphurous acids, or sulphides from which sulphuric acid may be formed. These substances dissolve gold in the presence of boiling nitric acid. A small quantity of silver is kept in solution, in the stock of acid, so that chlorine or any other haloid, if present, would instantly be detected. It is sometimes stated that the acid must be free from nitrous fumes, but this is not necessary, as silver protects gold from the action of nitric and nitrous acids, and as soon as parting begins great quantities of nitrous fumes are generated.

When the flasks are used, 2 ozs. of nitric acid of specific gravity 1·2 are put into each flask and raised to boiling point. A cornet is then introduced and boiling continued for fifteen or twenty minutes (*i.e.*, for about ten minutes after nitrous fumes cease to be given off). Hot distilled water is then added, the solution of nitrate of silver decanted off, and the flask washed by filling with hot water, and decanting. Two ounces of hot nitric acid of specific gravity 1·3 is now poured in and the boiling continued for fifteen or twenty minutes, a piece of fireclay or capillary glass tube being added to prevent bumping; after which decantation and washing is twice performed. Another boiling with acid of specific gravity 1·3 is recommended by Chaudet with the object of dissolving out the last traces of silver and leaving the gold quite pure. This practice has been adopted by many assayers, but is useless and causes loss of gold. If any small particles of gold have become detached from the cornet, time must be allowed for them to settle before each decantation. After the last decantation the flask is filled with hot water, the top covered by a small porous crucible, and the whole is carefully inverted; the pure gold, which is of a dark brown colour and exceedingly fragile, falls through the liquid and rests in the crucible, the water which enters with it being afterwards poured off. The crucible is dried and then annealed at a red heat over gas or in the muffle, when the gold shrinks greatly, though still preserving its shape, hardens and regains its ordinary pale yellow colour. It can now be weighed.

When a platinum boiler is used the cornets are put on platinum pins, as at the Sydney Mint, or more usually into platinum cups, one of which is shown in Fig. 103. These cups are supported in a platinum tray (which holds 144 cups at the Royal Mint) and the whole lowered by a platinum hook into a platinum vessel containing about 60 ozs. of hot nitric acid. Some attention must be paid to the temperature of the acid. At the Royal Mint acid of specific gravity 1·26 (in which, however, a small quantity of silver is already dissolved) is used, and the temperature at the moment of introduction of the tray is from 90° to 100° C. At higher temperatures the action is so vigorous that the vessel may boil over. Formerly, when the ratio of 2·75 parts of silver to 1 part of gold was used, great care was necessary in putting the assay pieces into the acid. The temperature of 90° could not be departed from widely. If the acid was colder than this the cornets tended to break up, some pieces being usually detached in the boiling of 144 cornets, even if the temperature was only 1° or 2° lower than the correct point. If the temperature of the acid is as low as 85°, the cornets are not immediately attacked; they turn a coppery hue, and then bubbles of gas form slowly on them. In such cases (where the action does not begin within one or two seconds after entrance into the acid) cornets containing 2·75 parts of silver to 1 of gold all

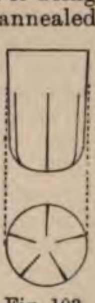
Fig. 103.

break up in acid of sp. gr. 1·26, and the assays are spoilt. Cornets containing 2 parts of silver to 1 of gold are less delicate. They can be put into acid of sp. gr. 1·26, even if it is cold, or at any temperature up to the boiling point, without showing any signs of breaking up. They can also be put into hot acid of sp. gr. 1·32 without injury, but are not safe in cold acid of the same strength. They break up in acid of sp. gr. 1·42, whether it is hot or cold. Cornets containing larger proportions of silver may not break up if the first acid is weaker, say of sp. gr. 1·2.

Boiling is kept up gently for about thirty minutes, and the tray is then withdrawn, drained, washed by dipping vertically in and out of a vessel of hot distilled water, drained again, and placed in a second platinum boiler filled with boiling nitric acid of specific gravity 1·32 (also containing a little nitrate of silver). In this the cornets remain for a period of about thirty minutes, when they are drained and washed as before, and are then ready for annealing.

The platinum tray is dried on a hot plate, and annealed over gas or in a muffle kept specially for the purpose, and free from lead fume, particles of bone-ash, &c. The temperature should be as high as possible, consistent with the safety of the cornets. They fuse at 1,064° C. (having the same melting point as pure gold), at which temperature the muffle appears orange-red. If annealed at a low temperature, the cornets are rough in texture, dull and fragile, being crushed easily between the finger and thumb. In this condition they adhere to the platinum, and, in detaching them, fragments are often left sticking fast inside the cups. If annealed properly the cornets are smooth, lustrous, and hard, showing signs of incipient fusion under a magnifying glass, and only yielding to considerable force exercised by the finger and thumb. Under these conditions they can always be detached from the platinum entire, and do not readily absorb moisture and gases from the air. By being annealed the cornets, which after boiling are very soft and fragile, and dark brown in colour, shrink, and harden, and regain the ordinary yellow colour of gold (*e*, Fig. 102).

Relative advantages of Parting in Flasks and in Platinum Boilers.—The use of platinum trays and boilers effects a great saving of time in decanting and washing, as one operation takes the place of as many as 144. If the standard of an alloy is unknown, so that it is not certain that the cornet will remain entire in the acid, it must, of course, be boiled separately, as, if one cornet in the tray breaks up, fragments may adhere to a number of others. The manipulation of the platinum tray is easier than that of parting flasks, and, in addition, the treatment of the cornets is more uniform, so that the correction afforded by the use of checks is more trustworthy. On the other hand, if platinum or palladium is present in an individual cornet, it imparts a straw-yellow or orange colour to the acid; but where a number of cornets are boiled together, it is obviously impossible to say from which the colour is derived, so that less information on the subject is obtained.

6. **Weighing the Cornets.**—The balance must readily indicate differences of 0·05 per 1,000, or $\frac{1}{40}$ milligramme, with a weight of $\frac{1}{2}$ gramme in each pan. The "checks" or "proofs" (*vide infra*) are weighed first, and their mean excess or deficiency in weight applied as a correction to all the cornets worked with them. This may be done by means of a light rider. The "weighing-in" correction (p. 472) is also allowed for, and the report is at once indicated by the marks on the weights without further calculation.

The weights are the $\frac{1}{2}$ gramme, which is stamped "1,000," and decimal subsidiary weights stamped 900, &c., down to 0·5. The stamped numbers denote the number of $\frac{1}{2}$ milligrammes ("millièmes") contained in the weight. Ordinary weights in the gramme system may, of course, be used, each milligramme corresponding to 2 millièmes in the assay system. The report finally made gives the number of parts (in millièmes and tenths) of pure gold in 1,000 parts of the alloy.

The weights usually supplied even by the best makers differ from the standard weights by varying amounts, which sometimes exceed 0·1 millième. These errors cannot in general be disregarded. In many sets of weights a cumulative error of 0·5 millième or more may be introduced. It is necessary to allow for the errors, or still better, to adjust the weights by carefully polishing them with rouge, or by finest emery, or by rubbing them on a ground glass plate, if they are too heavy, or by plating them with gold if they are too light. It is not advisable to gild weights unless they consist of platinum. Riders should be used instead of small weights. For further remarks on weights see above (pp. 458-459).

Auxiliary Balance for Weighing Cornets.—Mr. Robert Law, of the Melbourne Mint, has invented* an auxiliary balance which gives the approximate weight of the cornets without the use of any weights. The balance has a pan to hold the cornet attached to one end of the beam, and the other end is prolonged as a pointer, the position of which (when it comes to rest), with reference to a curved scale, denotes the percentage of gold present. The cornet is then transferred to the pan of an ordinary balance, and the necessary weights, already determined by the auxiliary balance within narrow limits, are placed on the other pan. For weighing large numbers of cornets of widely

* *Chem. Soc. Journ.*, May, 1896, p. 526.

differing weights, this balance is a great convenience, saving both time and wear of the balance and weights. It has been in use for some years at the Melbourne Mint.

Surcharge.—The gold cornet does not actually contain the whole of the gold present in the original alloy and nothing else. Gold is lost by (*a*) volatilisation; (*b*) absorption by the cupel; (*c*) solution in the acid. On the other hand, the cornet always retains (1) some silver; (2) occluded gases. The algebraical sum of these losses and gains is called the "surcharge," since the cornet usually weighs more than the gold originally present in the assay piece; if the reverse is the case, the work is less accurate. The various losses and gains are discussed in detail below.

Losses of Gold.—G. H. Makins* found gold and silver in the proportion of about 1 to 9 in the dust taken from a flue used only in gold and silver cupellation, but did not attempt to ascertain the percentage loss of gold by volatilisation. He also showed that large amounts of gold were dissolved by nitric acid in the course of assaying, and attributed the dissolution of gold to the presence of nitrous acid, but supposed that it would not be dissolved in the weaker acid, where nitrous acid was formed in larger quantities, owing to the protective action exercised by undissolved silver, which formed the positive element in the gold-silver couple. The fact that gold is actually dissolved in nitric acid, presumably as oxide, and remains dissolved even after dilution with water, was proved by A. H. Allen (*Chem. News*, vol. xxv. (1872), p. 85).

As the result of a number of experiments made by the author at the Royal Mint,† it was found that in the assay of standard gold (916·6 fine) with the ordinary surcharge of 0·4 to 0·8, the loss of gold is about 0·4 per 1,000, of which about 82 per cent. is absorbed by the cupel, 8 per cent. is dissolved in the acid, and the remaining 10 per cent., which is unaccounted for, is probably volatilised. These ratios, however, vary considerably, as a hot fire increases the loss by absorption, while by prolonged boiling in acid, and especially by annealing the fillets at a high temperature, the amount of gold dissolved in the acids is increased.

An increase in the percentage of copper in the assay piece is accompanied by an increase in the loss, as is shown in the following table which gives the relative surcharges obtained in the parting assay of gold-copper alloys of different standards; it is compiled from a number of results obtained at the Royal Mint:—

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* *Journ. Chem. Soc.*, vol. xiii. (1860), p. 77.

† *Ibid.*, vol. lxiii. (1893), p. 710.

Standard of Alloy.	Surcharge on the Assay Piece per 1,000.	Loss of Gold per 1,000 of Pure Gold.
916·6	+ 0·250	0·63
900·0	+ 0·225	0·65
800·0	− 0·075	1·00
666·6	− 0·2	1·20
546·0	− 0·7	2·20
333·3	− 2·8	9·30

The third column is obtained from the surcharge and the fineness in gold of the cornets obtained. The amount of lead used was eight times the weight of the alloy in the first three cases, and double this quantity for the last three. This table, as already explained, only gives relative surcharges and losses of gold; the absolute amounts vary with the treatment.

The influence of the temperature of the furnace on the surcharge is pointed out by Roberts-Austen.* The following table is compiled from the results of his experiments:—

Temperature of Furnace.	Composition of Alloy.	Surcharge.
(*a*) Slightly lower than usual, . .	Gold 916·7 Copper 83·3	+ 0·05
(*b*) Ordinary temperature, . .		− 0·10
(*c*) Slightly higher than usual, . .		− 0·37

The loss of gold was found to be 0·645 per 1,000 in series (*a*), and 0·723 part in series (*b*).

Rössler† has shown that the loss of gold in cupellation increases with the amount of lead used and decreases as the amount of silver is increased.

Silver Retained in the Cornet.—It has been found at the Royal Mint that after boiling in the first acid the average amount of silver retained is about 2·5 parts per 1,000. The amount left undissolved by the stronger acid varies with the length of time of its action. Under normal conditions with a surcharge of 0·6 to 0·8, the silver left in the cornets is from 1·0 to 1·2. If the acid continues to boil until its constant strength is reached, gold is dissolved to a considerable extent (0·5 per 1,000), while the proportion of silver present suffers little diminution, rarely falling below 0·6. If much gold is dissolved the results are not so uniform. When checks are used perfectly satisfactory results are obtained with surcharges of from + 0·2 to + 1·0 per 1,000 or even more.

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* Percy's *Metallurgy of Silver and Gold*, p. 275.

† *Dingl. Polytech. Journ.*, vol. ccvi., p. 185.

Effect of Varying the Proportion of Silver to Gold.—The views of various authors on the best proportion to be used have been given above (p. 472). The following are some of the results of a series of experiments made at the Royal Mint in the years 1904-5. In each case the results are the means of a number of closely concordant assays worked together under similar conditions. Except where stated otherwise, the cornets were parted by boiling in nitric acid for two half-hours, the first acid being of sp. gr. 1·26, and the second acid of sp. gr. 1·32. In all cases 0·5 grm. of gold, 0·045 grm. of copper, and 4 grms. of lead were used:—

Ratio of silver to gold, . .	2·71	2·50	2·00
	Per 1,000.	Per 1,000.	Per 1,000.
Surcharge, ...	+ 0·292	+ 0·208	+ 0·062
Loss of gold, ...	0·474	0·547	0·593
Silver retained by cornet, .	0·766	0·755	0·655

When the ratio of 2 parts of silver to 1 part of gold is used, it is seen that the amount of silver retained by the cornet is less, the loss of gold greater, and the surcharge less than when more silver is used.

If the surcharges are higher, the relation of the results remains approximately the same as shown below:—

Ratio of silver to gold, ...	2·75	2·00
	Per 1,000.	Per 1,000.
Surcharge—Series (*a*), ...	+ 0·608	+ 0·466
,, ,, (*b*), ...	+ 0·873	+ 0·705

If less than 2 parts of silver to 1 of gold are used, the surcharge begins to rise again, as follows:—

Ratio of silver to gold, .	2·17	2·00	1·90	1·75
Surcharge, per 1,000, .	+ 1·05	+ 1·03	+ 1·15	+ 1·60

If the ratio of silver to gold is lower than 1·75 to 1, the cornets are not properly parted by boiling in two acids in the ordinary way. By boiling for half an hour each in three acids of sp. gr. 1·26, 1·32, and 1·41 respectively, the following results were obtained:—

Ratio of silver to gold, .	2·00	1·90	1·75	1·50	1·25	1·00
Surcharge, per 1,000, .	+0·80	+0·80	+0·87	+1·52	+3·60	+533·0

Among other results, it was found that the uniformity of the assays was greater when the ratio of 2 to 1 was used than in the case of other ratios. This led to the adoption of the ratio for bullion assays at the Royal Mint.

Occluded Gases.—Graham[*] proved that cornets retained twice their volume of gases (mainly carbon monoxide) in occlusion after annealing. This amounts to two parts by weight in 10,000, and is reckoned as silver in the preceding paragraph. According to Varrentrapp, the gas retained varies with the temperature at which annealing takes place.

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[*] *Phil. Trans. Roy. Soc.*, 1866, p. 433.

Checks or Proofs.—Since the losses and gains detailed above are dependent on so many conditions, it is always necessary to subject check-pieces of known composition to the same operations as the alloys under examination. The use of checks in the Royal Mint was prescribed by law as early as the fourteenth century.* Standard trial plates (916·6 fine) were made and used for this purpose. If such trial plates contain copper, however, it is impossible to guarantee that they shall have absolutely the same composition throughout, so that it is better either to use trial plates consisting entirely of gold alloyed with silver, which are uniform in composition, or else to use pieces of pure gold, a corresponding amount of pure copper being added in order to make the assays absolutely comparable.† If 1,000 parts of pure gold were taken originally as a check, and the weight of the resulting cornet is 1,000·3, then 0·3 must be deducted from all the other results. No appreciable error is caused, provided that the standard of the alloys under examination is not much below 1,000. For example, if an alloy is 900 fine, the amount to be deducted may be assumed to be nine-tenths of 0·3 per 1,000 or 0·27, the error in deducting 0·3 being in this case one-tenth of the surcharge. Experiments show that the assumption made above is justified, and that the loss of gold is proportional to the amount of gold present if the composition of the assay pieces is similar. The errors introduced by the use of different weights are usually much greater, unless they are carefully adjusted (see above, p. 483).

In the foregoing paragraph it was assumed that absolutely pure gold was used for proofs. It is doubtful whether it has ever been obtained. The method of preparation of proof gold in use at the Royal Mint is given below. Since the assay of an alloy only gives the relative fineness of proof gold and the alloy, it follows that, if the proof gold is not quite pure, the amount found in the alloy will be in excess of the truth. If a sample of proof gold is less pure than the finest yet obtained, an allowance is made. Thus, if it is 999·9 fine, a deduction of 0·1 is made from all results of assays checked by it. This deduction is readily proved to be a very close approximation to the correct one.

Lastly, the "weighing-in" correction is applied. If the original weight taken was, say, 1,000·4 (recorded as + 4), it is sufficient to deduct 0·4 from the final weighing. In this correction 0·4 of alloy is reckoned as fine gold, but the error is inappreciable when alloys differing but little from pure gold are under examination. In the case of an alloy 900 fine, nine-tenths of the weighing-in correction must be added to or deducted from the weight of the cornet. If the alloy is 500 fine, one-half of the weighing-in correction must be applied, and in general if the weighing-in correction is x and the weight of the cornet is y, the correction to be applied in weighing out is $\frac{xy}{1,000}$.

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* *Mint Report for 1873*, p. 38.

† Roberts-Austen and Rose, *Proc. Roy. Soc.*, vol. lxvii. (1900), p. 105, and vol. lxxi. (1902), p. 161.

Preparation of Pure Gold.—The purest gold obtainable is required for use as standards or check pieces in the assay of gold bullion. The following method of preparing it is now in use at the Royal Mint. Gold assay cornets from the purest gold which can be obtained are dissolved in nitrohydrochloric acid, and the excess of nitric acid expelled by evaporation with additional hydrochloric acid on a water bath. The blackish-red fused product, smelling of chlorine and consisting chiefly of $AuCl_3 . HCl$, or $HAuCl_4$ (chlor-auric acid), is then poured in a thin stream into a large glass vessel full of distilled water, and a solution of about 1 oz. of gold in each pint of water (1 gramme of gold in 20 c.c. of water) is formed in this way. After vigorous stirring the solution is left to settle, and at the end of about a week, the whole of the precipitated chloride of silver will have subsided to the bottom. The progress of the subsidence is easily watched. The particles fall at the rate of about 3 or 4 inches per day. The clear bright supernatant liquor is now removed by a glass siphon, and diluted to about 1 oz. of gold per gallon of water (1 gramme to 150 c.c.). No further subsidence of silver chloride takes place, even if the diluted liquor is left undisturbed for three months, and the gold may be at once precipitated. Oxalic acid is often used, and is stated by Krüss to be the best precipitant if platinum is present. If platinum is absent, as in good cornet gold, carefully purified sulphurous acid gas is a more convenient precipitant than oxalic acid, and may be substituted for it without any ill effects. Platinum is not precipitated from its solution as tetrachloride by sulphurous acid, and tellurium, though reduced by sulphurous acid at the same time as gold, would not be present in the solution under the conditions named. Sulphurous acid acts in a few minutes in cold solutions, but if oxalic acid is used, it is better to warm the solution and to leave it to stand for three or four days.

After the precipitated gold has settled, the acid solution is siphoned off and the gold transferred to a large flask and repeatedly shaken with cold distilled water, closing the mouth of the flask with a watch-glass. The gold is then boiled for a few days in an apparatus resembling Soxhlet's fat separator. The gold is wrapped in well-washed muslin, and placed in the glass tube, A (Fig. 104). Water distils from the vessel, D, and the steam passes through the tube, E, and condenses in a condenser, placed above B, and drips on the gold. The water collects in A, and siphons over into D through the tube, C. The use of this washer was suggested by J. Phelps.

The gold is then turned out into a porcelain basin, covered over, dried, melted in a clay pot, and poured in an iron mould, which is neither smoked nor oiled, but rubbed with powdered graphite, and then brushed clean with a stiff brush. The ingot is cleaned by brushing and heating in hydrochloric acid, dried, and rolled out. The rolls must be clean and bright, and free from grease. The surface of the rolled gold plate is again cleaned by scrubbing with fine sand and ammonia and also with hydrochloric acid, and is scraped with a clean knife before being used as proofs in bullion assay.

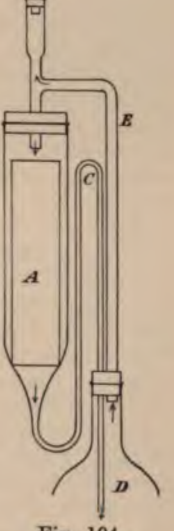

Fig. 104.

The amount of gold prepared at one time is usually from 40 to 50 ozs., but only 20 ozs. can be washed in a single flask of a capacity of 160 ozs. or $4\frac{1}{2}$ litres, and only about 10 ozs. treated at one time in the Soxhlet apparatus. The addition of hydrobromic acid to the clear solution of gold chloride has been tried without throwing more silver out of solution or affecting the quality of the gold. An additional precaution is to remove the gases taken up by the gold during the process of melting by heating it to redness *in vacuo*, but the consequent improvement in fineness, if any, cannot be detected by assay.

Limits of Accuracy in Gold Bullion Assay.—Attention may here be drawn to the errors introduced by the lack of delicacy of the very finest assay balances in ordinary use. It has elsewhere been shown by the author * that by weighing in the way indicated above, errors not greater than 0·15 per 1,000 may be introduced. It is, therefore, clear that this amount represents the limit of accuracy when such balances are used. By weighing correctly to 0·01 per 1,000, however, and performing all other operations with scrupulous care, then in the determination of gold in high-standard alloys of gold and copper, or of gold, silver, and copper, whether pure or contaminated by small quantities of lead, bismuth, zinc, antimony, nickel, and some other elements, the error does not exceed ± 0·02 per 1,000, if the mean of three results is taken.

Parting by Sulphuric Acid.—The use of sulphuric acid of

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* *Journ. Chem. Soc.*, vol. lxiii., pp. 700-714.

66° B. instead of nitric acid for parting is recommended by some assayers on the ground that the losses of gold by dissolution in nitric acid are variable, while sulphuric acid does not dissolve gold. The inconveniences suffered by the use of sulphuric acid are that (1) lead and platinum are left undissolved by it; (2) violent bumping of the liquid occurs during ebullition; and (3) sulphate of silver is not very soluble in water, and the washing is consequently done with dilute sulphuric acid. However, it is stated that less silver is left undissolved in the cornets than in the parting by nitric acid if the proportion of gold to silver is between 1 : 2 and 1 : 3.

Preliminary Assay.—If the composition of an alloy is quite unknown, a preliminary assay is necessary in order to determine the right quantities of silver, copper, and lead to be added. This determination may be made by the touchstone, by considerations of the colour and hardness of the alloy, or by cupelling 2 grains of it with 6 grains of silver and 30 grains of lead, and parting the button in a flask. Simple cupellation with lead gives satisfactory results if silver is absent or insignificant in quantity; according to Frémy this method, in which parting is dispensed with, is accurate to 3 millièmes if carefully performed with proofs.

Assay of Gold by means of Cadmium.—Balling has shown[*] that cadmium may be substituted for silver in the operation of parting. The ½ gramme of gold alloy is placed in a porcelain crucible in which a little fragment of potassium cyanide has been previously fused in order to protect the metal from the air. Cadmium is then added in the proportion of 2½ to 1 of gold. If silver is present in addition, the combined weight of cadmium and silver must be 2½ times that of the gold. The whole is fused and then cooled and plunged into hot water to clean the button, which is then crushed, parted in nitric acid (specific gravity 1·3), boiled in water for some minutes, dried and weighed. The silver, if any, can be estimated by precipitation as chloride from the acid solution. By this method the losses of gold and silver incidental to cupellation are entirely avoided. A similar method, employing zinc in place of cadmium, had previously been recommended by von Jüptner.[†]

Alloys of Gold, Silver, and Copper.—These may be assayed by the method just given, the copper being estimated as difference; or the gold may be estimated as usual, and other assay pieces cupelled with enough lead to remove all the copper. The buttons thus obtained contain silver and gold only, and the proportion of silver is found by difference. The method of double cupellation, by which the button of silver and gold is weighed and then subjected to inquartation and parting, is less accurate.

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[*] *Oestr. Zeitsch. für Berg. und Httnwesn.*, 1879, p. 597.

[†] *Zeitsch. für Anal. Chem.*, 1879, p. 104.

The cupellation designed to remove the copper is made with less lead than the quantities given on p. 473. If little gold is present, half the amount of lead there given is used, with increasing proportions as the amount of gold present increases. The temperature of cupellation must also be lower than for gold, approximating more to that used for silver. Proofs of similar composition must be used and the operations require much practice before the necessary skill is acquired. If less than 33 per cent. of gold is present originally, the alloy may be parted at once without cupellation and the silver estimated by weighing as chloride.

Effects of the Presence of other Metals on Gold Bullion Assay.—The effects on cupellation are the same as those given under ore assay (p. 452). In general, if a scoria is formed owing to the presence of large quantities of antimony, arsenic, cobalt, nickel, iron, tin, zinc, or aluminium, there is a loss of gold. The alloy should in that case be scorified with lead as a preliminary to cupellation. If mercury is present gold is carried off in the form of spray, and lost as the mercury boils off. The presence of tellurium is sometimes indicated by the formation of numbers of minute beads of precious metal dispersed over the surface of the cupel. Tellurium compounds are best analysed in the wet way (see p. 495).

If members of the platinum group are present they remain undissolved by the parting acid, and hinder the solution of the silver, and the assay is consequently rendered unreliable. The treatment of the alloys is discussed below (p. 493). The effects of the presence of small quantities of various metals on the surcharge in the ordinary parting assay is shown below. The table is the result of experiments made by the author.* The presence of 5 per cent. bismuth does not affect the surcharge. All assay pieces contained 1,000 parts of gold, 2,500 parts of silver, and 91 parts of copper, other metals being added in the proportions indicated.

Metal added.	None.	Antimony, 50 parts.	Zinc, 70 parts.	Tellurium, 33 parts.	Iron, 50 parts.	Nickel, 50 parts.
Surcharge,	+ 0·42	+ 0·28	+ 0·35	+ 0·07	+ 0·27	+ 0·22

These differences indicate the necessity of employing special checks containing these metals if such be present in the alloys.

ASSAY OF VARIOUS GOLD ALLOYS.

It is often impracticable to apply the ordinary parting assay to the examination of low-standard alloys of gold with other metals. These are then tested by various other methods, of which a summary is given below, the alloys being grouped in four series for convenience:—

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* *Journ. Chem. Soc.*, vol. lxiii. (1893), p. 706.

A. Alloys requiring scorification.
B. Amalgams.
C. Alloys containing members of the platinum group.
D. Tellurium compounds.

A. Scorification of Alloys.—Alloys of *arsenic* or *antimony* are reduced to a fine powder and scorified with thirty parts of lead and a half part of borax. If the slag becomes pasty towards the end of the operation more borax is added, a little at a time. If the lead button obtained is hard, a second scorification is necessary, with the addition of more lead. There is some loss of gold in the slag.

Iron or Manganese Alloys.—The operation is tedious and difficult with these alloys, as they are difficult to fuse, having higher melting points than pure gold,* and the oxides of iron do not form easily fusible compounds with the litharge. An extremely high temperature is required; ten parts of lead, one of borax, and one of silica usually suffice.

Cobalt and Nickel.—Twenty parts of lead are used, but no borax at first, so that the oxidation of the nickel may not be hindered. A very high temperature and the subsequent addition of two parts of borax are necessary. Several successive scorifications are required as nickel and cobalt are difficult to oxidise.

Zinc.—Oxide of zinc does not form a fusible mixture with litharge, and the slag is only rendered pasty by borax, unless it is added in large quantities. Gold is lost in the slag, but the loss is minimised by slagging off the zinc as rapidly as possible. Use fifteen to twenty parts of lead and two to three parts of silica, with a little borax.

Tin.—Twenty parts of lead are required; oxide of tin is rapidly formed, but the slag is not easily fusible. Large amounts of borax are necessary, or still better, borax mixed with potash which forms a fusible stannate with SnO_2.

Aluminium.—Alloys containing this metal cannot be assayed by scorification and cupellation. As soon as fusion takes place in the muffle, aluminium floats to the top of the bath, being of low density, and is rapidly oxidised, producing alumina which forms an exceedingly infusible scoria not easily removed by litharge. The production of the latter, moreover, is checked by the scum.

Mr. Edward Matthey observes† that the removal of aluminium by digestion in hydrochloric acid, and collection of the residual gold, does not yield satisfactorily results. The process he recommends is as follows:—Accurately weighed portions of 50 grains each of the alloys are fused with litharge, under a flux of potassium carbonate and borax with a small proportion of powdered charcoal, and the resulting slag re-fused with a further small quantity of litharge and powdered charcoal. The lead buttons containing all the gold (the aluminium having combined with the fluxes employed) are cupelled, and the resulting gold cupelled with silver and parted with nitric acid in the usual manner. The assays must be worked with checks or standards of fine gold and pure aluminium.

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* Frémy, *Ency Chim.*, T. iii., L'or, p. 147.

† *Phil. Trans. Roy. Soc.*, 1892, p. 643.

In the majority of the preceding cases it is better to analyse the alloys by wet methods (see p. 496).

B. Amalgams.—The alloy is placed in a weighed porcelain crucible and gradually heated so as to drive off the mercury. After the greater part of the mercury has been driven off the temperature is raised to a full red heat which is maintained for half an hour. About 0·1 per cent. of mercury still remains in the gold after this treatment and can only be completely removed by cupellation and parting. Checks must be used, as the loss of gold in the operation may amount to 1 per 1,000.

A better method is to dissolve the mercury in nitric acid mixed with an equal volume of water. A porcelain crucible or basin should be used and gentle heat applied. If the action is not too violent the gold does not break up, but is left as a spongy coherent mass. It is washed, first with nitric acid, and finally with water, and ignited. Mercury may be made use of to collect very finely-divided precipitated or parted gold into a coherent mass, the mercury being removed by nitric acid.

C. Platinum Group.—Cupellation must be performed at a higher temperature than usual, and the iridescent bands are seen to remain longer, although they are less numerous.

(1) *Platinum Gold Alloys.*—The button obtained by cupellation is dull and crystalline. If the alloy contains as much as 7 or 8 per cent. of platinum the cupellation proceeds slowly, brightening is only obtained at a very high temperature and the button appears flattened, and has a rough crystalline surface and a grey colour. If more than 10 per cent. is present, brightening does not occur at all, and the other features just mentioned are more strikingly exhibited. On parting, the platinum is partly dissolved with the silver, but the assay-piece must be boiled in acid for a long time, and the parting is incomplete. A second and even a third parting may be necessary. Unless proofs are made up of similar composition the results are not satisfactory if more than 1 or 2 parts of platinum are present per 1,000 of alloy.*

According to results obtained in the laboratory at the Royal Mint by Prof. Stansfield, it is impossible to free buttons containing platinum from lead at the ordinary temperatures attainable in a muffle. It is necessary to employ the oxyhydrogen blowpipe for the purpose, with the result that the losses are irregular. In the cupellation of pure platinum, buttons weighing from 0·002 to 0·01 gramme retain about 10 per cent. of their weight of lead, and those weighing from 0·04 to 2 grammes retain about 33 per cent.

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* According to Spiller (*Proc. Chem. Soc.*, 1897, No. 180, p. 118), platinum is but little attacked by nitric acid in the presence of silver, only about 1 per cent. being dissolved.

If parting is effected by boiling in concentrated sulphuric acid instead of in nitric acid, almost all the platinum remains undissolved with the gold.

Chaudet* recommends the following proportions of silver for this purpose :—

Ratio of Platinum to Gold.	Ratio of Silver to Platinum and Gold together.
Less than 1 to 10	2 to 1
1 ,, 1	1·5 ,, 1
More than 10 ,, 1	1·25 ,, 1

The reason for the reduction, shown in the table, in the amount of silver when the platinum is increased is that, according to Chaudet and Haindl, considerable quantities of platinum are dissolved, and the error caused thus is corrected by leaving some silver undissolved in the cornet. The losses of platinum, however, are very irregular, and the method is not a good one. The reason for the variation is probably the lack of homogeneity in the buttons and prills, as gold-platinum alloys show a considerable amount of liquation.

The platinum is not dissolved by nitric acid if zinc or cadmium is substituted for silver.

Mr. Edward Matthey recommends the following method for alloys of platinum 900, and gold 100 parts :—50 grains each are taken of the alloys and treated with an excess of nitrohydrochloric acid which gradually dissolves the whole. The resulting solutions of platinum-gold chloride are then evaporated nearly to dryness to drive off the free acid and diluted with distilled water to about 20 c.c., a degree of strength ascertained by experiment to be the best for the precipitation of the gold. The metallic gold is thrown down by means of crystals of oxalic acid, and is carefully washed, dried, and weighed. It is necessary to use checks.

The estimation of platinum in its alloys with gold and silver has also been studied by M. Forest, assayer at the French Mint.†

(2) *Palladium-Gold Alloys.*—The palladium is dissolved in parting if the weight of silver is at least three times that of the

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* *L'art de l'essayeur*, Paris, 1835.

† 6ème *Rapport par l'Administration des Monnaies et Médailles*, 1901, p. xxix.

gold, yielding an orange coloured solution. Matthey recommends double parting. Separation may also be effected by fusion with six to eight parts of potassium bisulphate, and dissolving out the dark brown palladious sulphate by boiling water.* A second fusion is usually necessary to render the gold residue quite pure.

As in the case of platinum, palladium may be separated from gold by dissolving the alloy in aqua regia, evaparating to dryness, taking up with water and precipitating the gold by oxalic acid from a very dilute solution : the palladium remains in solution.

(3) *Rhodium- and Iridium-Gold Alloys.*—Iridium, if present, always sinks to the bottom of the cupelled button as it is very dense (specific gravity = 21·38), and is not usually fused at the temperature of the muffle, but occurs in the state of fine black crystalline particles. Hence, when the button is rolled into a cornet with the lower face outwards (p. 480) iridium occurs as black sooty spots or streaks which are seen by a lens to fill up depressions in the surface of the gold.

Both rhodium and iridium are almost insoluble in aqua regia. If gold alloys containing both of them are parted in the ordinary way with nitric acid, only a small quantity of rhodium goes into solution with the silver. The residue consisting of the gold, the iridium and most of the rhodium may then be attacked by dilute aqua regia when the gold is dissolved together with only traces of the other metals. These may be separated by evaporating the solution to dryness and heating to dull redness, when the reduced metals being no longer alloyed may be completely separated by dissolving the gold in aqua regia.

According to d'Hennin,† iridium may be separated from gold by fusing it with fluxes, the charge being made up as follows :—

Iridic gold,	12·5	grains.
Sodium arseniate,	3	,,
Black flux,	18	,,
Ordinary flux (consisting of borax, cream of tartar, charcoal, and litharge),	20	,,

The iridium forms a speiss with the iron and the arsenic, and the lead button formed at the bottom of the fused mass contains all the gold.

The estimation of osmium, iridium, and ruthenium in gold bullion has also been studied by Riche, Leidié, and Quennessen,‡ assayers at the French Mint.

D. Tellurium Compounds.—These must be treated by wet methods. The alloy may be dissolved in nitrohydrochloric acid, and the solution containing both gold and tellurium evaporated with a large excess of hydrochloric acid until no more chlorine is given off, when both gold and tellurium are readily precipitated by a current of sulphur dioxide gas. On attacking the precipitate with dilute nitric acid the tellurium is dissolved in the state of tellurous acid, and the gold residue may be dried and weighed, and its purity ascertained by inquartation and parting. The greater part of the tellurium may be removed from gold-tellurium alloys by boiling in nitric acid, and the residue can be cupelled and parted with very little loss of gold.

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* H. Rose.

† *Dingl. Polyt. Journ.*, vol. cxxxvii., p. 443.

‡ *8ème Rapport par l'Administration des Monnaies et Médailles*, 1903, p. xxix.

Wet Methods of Assay of Gold Alloys, Compounds, &c.—Assays or complete analyses of gold bullion, natural minerals, &c., can be made by the ordinary chemical methods given by Fresenius, Crookes, and others. From 1 to 5 grammes of bullion are usually enough, but a much larger amount is necessary if the alloy is nearly pure gold. M. Forest[*] takes 300 grammes of gold bullion when examining it for small quantities of impurities. In general the residue left after prolonged action of nitric or sulphuric acid is not sufficiently pure to weigh as gold, and complete solution in aqua regia is usually necessary. From the solution the gold may be precipitated by (*a*) ferrous sulphate, (*b*) sulphurous acid, (*c*) oxalic acid, (*d*) sulphuretted hydrogen, (*e*) ammonic sulphide, followed by the addition of hydrochloric acid. The following remarks may be of value in aiding the chemist in his choice of a precipitant in any particular case.

Nitric acid must always be expelled from the solution by warming with successive additions of hydrochloric acid. The acid solution must not be heated too strongly or loss of gold chloride by volatilisation occurs. Some other chlorides escape more freely. Ferrous sulphate and sulphurous acid act well in strongly acid (HCl) solutions; oxalic acid, sulphuretted hydrogen, and ammonic sulphide best in presence of small quantities of HCl.[†] The solution should be dilute (say 1 part of gold in 300 of water), so that other metals may not be carried down by the gold. Sulphate of iron gives a very finely-divided precipitate which is difficult to wash by decantation without loss; precipitation is slow in cold solutions. Oxalic acid causes plates and scales to form which are readily washed and are very pure; it acts best in boiling liquids, but a temperature of 80° for forty-eight hours suffices; in the cold or in the presence of much hydrochloric acid or alkaline chlorides the action is very slow and partial; a large excess of the precipitant must be present. Oxalic acid is used for solutions containing metals of the platinum group, which are not precipitated by it. Alkaline oxalates act better than the free acid.

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[*] 9ème *Rapport par l'Administration des Monnaies et Médailles*, 1904, p. xxxi.

[†] H. Rose.

Sulphurous acid is an excellent precipitant for most solutions. It acts rapidly and completely in the cold, and does not readily precipitate other metals, except tellurium. Sulphuretted hydrogen is used in the absence of all other metals whose sulphides are insoluble in hydrochloric acid.

In all cases careful consideration must be given to the nature of the base metals present, and the precipitant which will not render any of them insoluble must be selected.

Other Methods of Bullion Assay.—Among methods which have been proposed at various times, and which may still be of service occasionally in particular cases, may be mentioned:—(1) The trial by the touchstone (a method formerly more extensively used by jewellers than at present); the assay by means of considerations as to (2) the colour, and (3) the density of alloys; (4) spectroscopic assay. A brief description of each of these methods is appended.

1. **Trial by the Touchstone.**—This is the oldest method of assay. It consists in rubbing the gold bullion to be tested on a hard smooth stone, and comparing the appearance and colour of the streak with those made by carefully prepared needles of known composition. The effect of the action of nitric acid and dilute aqua regia on these streaks is also noted. Touchstones usually consist of Lydianstone or of silicified wood, and black or dark green stones are best. Only alloys of gold and copper or of gold and silver can be thus tested. The trial is more sensitive for alloys below 750 fine than for higher standards. The amount of gold in alloys between 700 and 800 fine can be determined correct to 5 parts per 1,000.

2. **Colour and Hardness of Alloys.**—These properties form a guide to the composition of copper-gold alloys, an increase of copper corresponding to a heightening of the colour and an increase of the hardness as tested by shears or a knife. On heating the alloy to redness in air, the degree of blackening of the surface is a further indication of the percentage composition, if compared with plates of known fineness.

3. **Density of Gold-copper Alloys.**—The determination of the fineness of these alloys by taking their densities was investigated by Sir Wm. Roberts-Austen at the Royal Mint in 1876.* He showed that the densities found by experiments were nearly equal to those obtained by calculation on the assumption that the union of the two metals was accompanied neither by contraction nor expansion. The alloys examined ranged from 860 to 1,000 fine, and were made into discs which were all compressed to the same extent. The conclusion arrived at was that the fineness of large masses of gold can be deduced from their densities correct to $\frac{1}{10000}$ part. In the case of individual coins the results are only approximate.

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* *Seventh Annual Mint Report*, 1876, p. 41.

4. **Assay by means of the Spectroscope.**—This method of determining the composition of gold-copper alloys was investigated by Lockyer and Roberts-Austen.[*] The spectrum of pure gold was shown to be altered by successive additions of copper, and near the English standard (916·6 fine) a difference of 2 or 3 parts in 10,000 in composition could be readily detected, but the amount of metal volatilised is so small that it cannot be made to represent with certainty the average composition of the mass, which is never perfectly homogeneous.

The detection of traces of gold in alloys or ores by means of the spectroscope, though sometimes attempted, is not remarkable for its delicacy or certainty. The method of procedure[†] is to dissolve the auriferous material in aqua regia, evaporate off the nitric acid, and pass induction sparks through the surface film of liquid, when the spectrum shows some narrow bands and some nebulous bands. The latter only are seen if a drop of the solution is placed in a Bunsen flame. The method may sometimes be useful when complex minerals are being examined. Mr. Cupel has shown that $\frac{1}{4000}$ of a milligramme of gold will show a spectrum if the spark be passed through a weak solution of the pure metal. But when operating on a slip of alloy formed of

Silver, ...	708
Copper, ...	254
Gold, ...	38
	1,000

the spectra of copper and silver alone were visible. In an alloy of gold and copper containing from 200 to 250 parts in the 1,000 of the precious metal, the gold spectrum is barely visible. On the other hand, in an alloy of gold and copper containing traces only (·01 per cent.) of the latter, the copper spectrum was distinctly shown. Alloys of gold are rarely so perfectly homogeneous that the particles of metal volatilised and giving the spectrum represent the whole mass.

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[*] *Phil. Trans. Roy. Soc.*, vol. cxliv., part ii., p. 495, and *Mint Report for 1874*, p. 38.

[†] Frémy, *Ency. Chim.*, T. iii., L'or, p. 134.

CHAPTER XXI.¶

STATISTICS OF GOLD PRODUCTION.

Cost of Production of Gold.—In view of the fact that the value of money is measured in almost all gold-producing countries by the metal itself, it would be of special interest to estimate the average cost per ounce of its production. In attempting this, however, the difficulty is encountered that a large amount of gold is even now extracted in small mills or by individuals, particularly in the case of placer deposits, and the exact cost is frequently a matter of doubt to the proprietors themselves. Moreover, gold mining companies are often reticent as to their costs, although the advantages of this course of action to the proprietors, as distinguished from the managers, are not easy to understand. By certain large companies, systematic accounts are published, and from these the following results are extracted.

At the Alaska Treadwell Company's Mill about 20,000 tons of ore, containing 2½ dwts. of gold per ton, are treated monthly by crushing and amalgamation, followed by concentration and chlorination of the sulphides. The cost (including both mining and milling) per ounce of gold extracted was £2 1s. 1d. in 1890-91, £2 4s. 8d. in 1891-92, and £2 8s. 6d. in 1904-5.* At the El Callao Mine, Venezuela, the cost per ounce was formerly as low as 25s., but rose to £3 10s. 9d. in 1891, and £3 1s. 8d. in 1892. The cost to the Montana Company was £3 10s. 6d. per ounce in 1891, and as much as £4 8s. in 1892. In some large hydraulic mining companies the cost was similar to that of the Alaska Treadwell Mine. Thus, at the La Grange Company's workings in 1874-76, the cost was £2 4s. 3d. per fine ounce of gold, and at the New Bloomfield Company's Claim the cost was £3 0s. 10d. in 1875, and £2 1s. in 1876 and in 1877. At the Homestake Mines, S. Dakota, which include seven mines producing over 200,000 ozs. of gold per annum, the cost per fine oz. was £3 18s. in 1901-2, £3 14s. in 1902-3, and £3 in 1904-5. The average cost of production in California was estimated by A. G. Charleton† at about £2 2s. 6d. per ounce. In South Africa, on the Witwatersrand, the cost of production has been greatly reduced during the last few years. In 1892 the cost of production per fine ounce of gold at those mills which were working more or less continuously seems to have been about £3 10s., and in 1895, for the mines situated on the outcrop of the Main Reef, Hatch and Chalmers estimated it at about £2 14s. In the year 1904, on the Witwatersrand field, 8,058,295 tons of ore were milled, yielding 3,653,794 ozs. of gold from all sources, the average yield being 38·46s., or 9·05 dwts. per ton. The dividends declared during the year amounted to £3,855,312, or 24·84 per cent. of the value of the gold. In the first nine months of 1905, the output was 3,487,088 ozs., from 8,218,019 tons, and the dividends declared amounted to £2,478,422. This would give the cost of the gold won as £3 3s. 4d. per ounce in 1904 and £3 10s. 9d. in 1905, including the tax of 10 per cent. on profits. In Mysore, where fuel is costly and the climate bad, the cost was formerly about £1 18s. 10d. per ounce of gold extracted.* At the placer mine at Saint-Elie in French Guiana, which produces about 1,600 ozs. of gold per month, the cost of production was £2 1s. 6d. in 1887, and at the Siberian placers the cost is £1 17s. 2d. at Berezovsk, £1 17s. 8d. at Nijni-Tassnil, and £2 15s. 8d. at Tchernaia-Retchka. In seven of the leading mines at Kalgoorlie, West Australia, which produced about 531,500 fine ozs. in 1900, the mean cost per oz. was about £2 10s.†

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* The value of 1 oz. troy of fine gold is approximately £4 4s. 11·454d. or £4·2477273. The logarithm of this number is 0·6281566. The value of 1 kilogramme or 32·1507267 ozs. of fine gold is £136·56756, logarithm 2·1353475. In United States currency these values are—1 oz. $20.671452, 1 kilogramme $664.6025.

† *Trans. Fed. Inst. Mng. Eng.*, 1893, p. 217.

Judging from the published results generally, it would appear that the average cost of production of gold is somewhat less than £3 per ounce. It must be remembered, however, that the interest and amortisation (sinking fund) on the initial cost of discovering and developing the mine, and of the machinery used in both mining and milling is, in many cases, not included in the so-called cost of production. It has, moreover, been frequently asserted that if the working costs of unsuccessful mines, and the money spent in fruitless prospecting, and in the erection of mills unsuitable to the ore to be treated, were to be included in the cost of the production of the gold, it would be found that a heavy loss is annually experienced by the gold-mining industry.

Annual Production of the Gold Mines of the World.—The production of gold in ancient times cannot be closely estimated, but, judged from a modern standpoint, it was probably very small. In the middle ages, however, between the fall of Rome and the discovery of America, the production was far smaller than before, and Jacob observes that in this period "the precious metals were sought not by exploring the bowels of the earth, but by the more summary process of conquest, tribute, and plunder." Even after the exploitation of the New

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* *Ibid.*, p. 218.

† A. G. Charleton, *Mining and Milling in Western Australia*, pp. 210, 580.

World began, the output of gold was for many years much too small to satisfy the cupidity of the conquerors. The development of the mining industry was prevented by the ruin and destruction of the natives, and by the almost incessant irregular warfare waged against the Spaniards in America in the 16th century, first by the Dutch and later by the English. Fifty years elapsed after Columbus discovered America, before the annual production of gold reached £1,000,000, and even at the end of the 17th century Soetbeer estimates that it was only £1,500,000. The discovery and working of the rich Brazilian placers during the next half century raised the annual product to over £3,500,000 in the period 1740-1760 (Soetbeer), but as these deposits became exhausted, the output again fell off, and in the period 1810-1820 had again sunk to about £1,500,000 per annum. The gradual development of the Siberian placers was the main cause of the subsequent steady increase in production up to an average of £7,500,000 per annum in the period 1841-1850, and this was followed by a sudden rise consequent on the discoveries in California and Australia. The maximum output from the rich placers of these countries was reached in 1853, when the world's production of gold is estimated by Sir Hector Hay to have been £38,000,000. After falling to £21,000,000 in 1863 (Hay), the output remained nearly stationary until about the year 1888, when, from various causes mentioned below, the production again began to increase, and in 1899 reached £64,000,000. After a sudden fall to £52,000,000 in 1900 owing to the Boer War, the increase in production was resumed, and in 1904 the output was £72,000,000, the greatest amount on record.

The increase since 1888 is due (1) to the discovery and development of new districts, and (2) to the progress in the art of metallurgy. The goldfields in South Africa, in West Australia, on the Yukon, and at Cripple Creek in Colorado, have all been developed since 1888. The output of gold in South Africa was valued at £81,000 in 1887 and £17,000,000 in 1904; Cripple Creek had not been discovered in 1887, and produced about £3,300,000 in 1899; the Yukon district produced £140,000 in 1887 and £4,300,000 in 1899; while Western Australia produced £20,000 in gold in 1887 and £8,500,000 in 1904.

The increase of production due to improved metallurgical methods is difficult to estimate. Most of it must certainly be put down to the account of the cyanide process, although the development of the dredging industry is becoming an important factor. The increase of about £1,500,000 which took place in the Russian output between 1888 and 1897 is partly due to improvements in the methods of washing the gravels, and partly to the extension of the Siberian railway.

Speaking generally, it is clear that of the increase in the annual output of gold of some £50,000,000 which took place between 1887 and 1904, about two-thirds is due to the discovery of new fields, and one-third to the development of districts already known. The credit for having caused the latter part of the increase and a large part of the former is due to the cyanide process and to a number of minor improvements in metallurgical processes and in mining.

As regards the future production, it is obviously impossible to predict what will be the effect of the opening up of gold-fields as yet undiscovered. Even without these, however, it seems almost certain that the production will continue to rise rapidly for some years to come. It is expected that the South African output will be doubled in the course of a few years. Siberia may also contribute much more than at present to the world's production, and the improvements in methods of extraction which have been so marked of late years will doubtless help to swell the total. It is, therefore, no exaggeration to say that present indications point to an annual production of gold of £100,000,000 in a few years time.

The table on p. 503 gives details of the production of gold in various countries in the years 1894, 1902, and 1903. Full particulars of the production in 1904 are not yet available. The following estimate is approximately correct:—

GOLD PRODUCTION IN 1904.

Australasia,	4,195,307	fine ozs.
United States,	3,905,000	,,
Transvaal,	3,779,621	,,
Russia,	1,213,000	,,
Canada,	822,375	,,
British India,	555,600	,,
Mexico,	600,000	,,
Other countries,	1,800,000	,,
	16,870,903	,,

The product of each of the Colonies of Australasia is as follows[*]:—

	1902. Fine Ozs.	1903. Fine Ozs.	1904. Fine Ozs.
Western Australia,	1,871,037	2,064,801	1,983,230
Victoria,	720,866	767,297	765,600
Queensland,	640,463	668,546	624,917
New Zealand,	459,406	461,648	467,897
New South Wales,	254,435	254,260	269,817
Tasmania,	70,996	59,891	65,921
South Australia,	24,082	22,269	17,925
	4,041,285	4,298,712	4,195,307

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[*] *Thirty-fifth Annual Report of the Royal Mint*, p. 144.

GOLD PRODUCTION OF THE WORLD.

Country.	1894. Fine ozs.	1902. Fine ozs.	1903. Fine ozs.
Australasia,	1,989,300	4,041,285	4,298,712
United States,	1,937,500	3,870,000	3,560,000
Transvaal,	1,906,900	1,704,410	2,963,681
Mexico,	217,300	491,156	516,524
Canada,	53,000	1,032,161	911,118
Russia,	1,337,400	1,090,053	1,191,582
British India,	182,200	463,824	552,873
China,	290,900	422,401	354,334
Korea,	22,600	145,125	145,125
Colombia,	140,000	122,031	131,795
Guiana (French),	64,300	117,077	101,658
,, (British),	117,700	87,491	77,948
,, (Dutch),	28,000	15,577	18,183
Brazil,	107,300	101,584	110,016
Peru,	3,600	112,525	28,669
Central America,	22,800	96,842	90,716
Japan,	23 700	62,259	96,881
Austria-Hungary,	81,500	105,037	108,609
Venezuela,	39,000	20,985	4,087
East Indies (British),	...	49,686	56,899
,, (Dutch),	...	22,930	24,261
Ecuador,	3,300	9,675	13,272
Rhodesia,	...	172,899	196,682
Africa, West Coast,	1,000	19,352	59,981
Great Britain,	3,500	3,428	4,543
Abyssinia,	1,000	33,865	33,865
Madagascar,	1,000	38,484	55,150
Other Countries,	138,000	48,981	52,874
Chile,	22,400	32,262	32 262
	8,735,000	14,533,385	15,792,300

The product for the individual States in the United States for 1903 is as follows :—*

Colorado,	1,090,376	fine ozs.
California,	779,057	,,
Alaska,	416,738	,,
South Dakota,	330,243	,,
Montana,	213,425	,,
Arizona,	210,799	,,
Utah,	178,863	,,
Nevada,	163,892	,,
Idaho,	75,969	,,
Oregon,	62,411	,,
Other States,	38,227	,,
	3,560,000	,,

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* *Report of the Director of the Mint* (*United States*) *for 1904*, p. 48.

The table given on p. 503 is based on Government returns for countries included in the British Empire. The other amounts are taken from the *Reports of the Director of the United States Mint*. The production of most of the countries of South America, and of China, Japan, Korea, and the Malay Peninsula are estimated.

Consumption of Gold.—Many attempts have been made at various times to estimate the amount of gold used annually for purposes other than that of additional coinage. In 1831, Jacob* estimated the annual amount of gold "converted into utensils and ornaments" at about £4,500,000, while the production was under £2,000,000, without counting, however, imports from India and China, which were supposed to be considerable. In 1881, Dr. Soetbeer estimated the annual industrial consumption of gold in the world, after making deductions for old material employed, as amounting to 84,000 kilogrammes or £11,500,000, and in 1885 he put it at 90,000 kilogrammes or £12,300,000, against a production of £21,000,000. In 1891, the same authority gave the industrial consumption of gold added to the amount hoarded and that exported to the East as 120,000 kilogrammes or £16,400,000, against a production of £24,000,000. In 1894, Ottomar Haupt estimated the industrial consumption at only 280,000,000 francs or £11,200,000, but the exports to the East are not included in this amount. In the same year the Director of the United States Mint † estimated the industrial consumption at $50,177,300 or about £10,300,000, and for the year 1903, at $76,350,600 or about £15,700,000. Doubtless the industrial consumption of gold is increasing in amount with the increase of population and wealth, but nevertheless it is fair to assume that no such rapid growth has taken place of late years in consumption as has been already pointed out to have taken place in production.

Taking the annual consumption of gold, including that exported to the East, as being, on the basis of the estimates given above, from £15,000,000 to £20,000,000, it would appear that the amount of gold added to the world's coinage, or retained as reserves in the form of bullion, was about £5,000,000 per annum previous to the year 1890, and gradually increased until 1904 when it was probably about £50,000,000. It was estimated that the total available stock of gold in the world (1894) was $3,965,900,000, or about £813,500,000, and on December 31, 1903, the stock was estimated at $5,685,700,000, or about £1,166,000,000. The addition to the stock was, therefore, £352,500,000 in ten years, or at the rate of 4·34 per cent. per annum. With regard to the effect which this increase may have on prices and on commercial prosperity, it is to be observed that in the years succeeding 1848, when the available stock of gold is estimated to have been less than £300,000,000, an annual addition (exclusive of the industrial consumption) was made to the available stock of about £20,000,000, or 6·7 per cent. of the total amount. This addition is usually supposed to have stimulated trade and inaugurated a long period of commercial prosperity. Jevons estimated that the purchasing power of gold was depreciated by from 9 to 15 per cent., and that it was only kept from falling lower by the absorption of vast quantities of gold for use in currency by several countries which had previously mainly employed silver or paper.

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* *History of the Precious Metals*, vol. ii., p. 322.

† *Production of the Precious Metals*, 1893, p. 53. *Report of the Director of the Mint*, 1904, p. 48.

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